MINERALS

ECONOMIC & REGIONAL DEVELOPMENT AGREEMENT

Canada

Manitoba

MG-13097A

TITLE: TO DEVELOP SULPHUR REDUCING MINERAL BENEFICATION TECHNIQUES AND PERFORM SULPHUR EMISSION TESTS AT INCO SMELTER, THOMPSON, MANITOBA

Final Report (#22)

By:

R. M. Perchaluk

Work on the project was conducted under the auspices of the Canada/Manitoba Mineral Development agreement.

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MICROMEDIA

FINAL PROGRESS REPORT

DSS File No. 06SQ,23440-5-9030

General Summary of Project Initiatives and Costs

Test work during April - July, 1989 completed the final two items of concern, which were column flotation testing of Pn/Po separation on scavenger cleaner feed and Pn/Po separation of scavenger cleaner feed using diethylene triamine (DETA). This work was conducted at Mill Process Technology expense as the $338,864 budgeted with Canmet was depleted as of March 31, 1989.

Dr. R. G. Orr continued his study on possible processed to treat Refinery sulphide anode slimes (SAS) in Thompson and issued several reports on the subject. Dr. Orr is preparing an overall report on the Manitoba Division future SO2 handling strategy.

The Mill pyrrhotite study was audited on August 30th by Lawrence A. Melis (Melis Engineering Ltd.) and Lois H. Epp (Brystra Consultants). The project personnel interviewed were R. M. Perchaluk and R. G. Orr.

Project Overview

Since test work conducted on the project was reported in detail in reports 1 - 21, it is felt that a project summary at this point would suffice as a final report.

The objective of the test work was to reduce SO2 emissions in the Thompson Smelter by developing mineral benefication techniques to reduce the sulphur recovery to the Smelter feed while minimizing metal losses to Mill tailings.

The scope of work covered; (A) mineral benefication which looked at mineralogical examinations and lab, pilot plant, and plant demonstrations, (B) Smelter energy balance and processing operations. Since Dr. R. G. Orr will be summarizing section B in his final report to the government, I will not elaborate on his test work in this report.

The study package was divided into 7 phases, which are as follows:

1. Optimize regrind - liberation in the scavenger cleaner circuit
2. Magnetic separation
3. Alternative reagents
4. Column flotation

5. Current ("A"-Circuit) optimization

6. Rougher concentrate Pn/Po separation

7. Smelter energy balance

The test work was carried out by Thompson Mill Process Technology personnel under project coordinator R. M. Perchaluk. Considerable test work was also carried out by Inco's Sheridan Park research laboratory under Dr. Gordon Agar's supervision. The Smelter energy balance study was coordinated by Dr. Ron G. Orr.

It should be noted that the test work was done exclusively on Thompson underground type ores which are characterised by high content magnetic (monoclinic) pyrrhotite gangue. Although it is known that by 1995 almost 50% of the Manitoba Division feed will be ultramafic related, no test work was done on this type of material since representative samples were not available. Dr. Orr did, however, do a study on previously processed Pipe Lake Open Pit ore and made some predictions on what we might expect.

The project challenge was to determine the best means of meeting the proposed 1994 Manitoba Division SO2 emission limits (220 K tonnes/yr) at a net product production rate of 140 million lbs/yr. For the Thompson type ores this would require a sulphur rejection rate of around 62%. Projections based on previous studies suggested that Mill nickel losses would increase by an added 10 million lbs per year (86% nickel recovery vs. current 91.5% recovery).

Following is a summary of test results related to each phase of the study.

1. Optimize Regrind/Liberation in the Scavenger Cleaner Circuit

Since almost 75% of the flotation circuit pyrrhotite is recovered to the scavenger side of the flowsheet, it should not be surprising that the majority of our effort was spent on Pn/Po separation on this material.

A major mineralogical study was conducted on key flotation streams and the results were reported in progress report #13 issued in February, 1987. The study reported that the pyrrhotite in Thompson type ores was almost 100% monoclinic (magnetic) in nature. Furthermore, the feed to flotation was well liberated with locked particles occurring in only the +100 mesh (149 micron) fraction. Pyrrhotite as a substantial Ni carrier was present in significant amounts only in the scavenger and scavenger cleaner circuits where it accounted for between 18 - 25% of the nickel. Generally speaking, liberation was not a problem.

However, test work (plant scale) was scheduled to evaluate the benefits of scavenger cleaner feed regrind using the #5 regrind mill. Conclusive results were never obtained as test work had to be curtailed on the several tests tried due to Smelter heat balance problems. On one occasion sulphur rejection of 56-58% was achieved at nickel recoveries of >90% using regrind.

Gordon Agar (J.R.G.R.L.) also found that he could achieve a 56% sulphur rejection level by treating scavenger cleaner feed with high sodium amyl xanthate (SAX) rates and adding lime in place of soda ash. These results were published in the 1987 progress reports.

Conclusions

Sulphur rejection levels of 55 - 57% can be achieved with current scavenger circuit flowsheet using soda ash (although higher SAX levels and lime would also work). Since some nickel is locked with the +100 mesh fraction, regrind of scavenger cleaner feed should be beneficial for increased Ni recovery at higher sulphur rejection levels, but project test work did not conclusively prove this. At 55-57% sulphur rejection, nickel recoveries of 90% or better are achievable assuming T-1 type ores.

Future Study

1. The majority of the pentlandite mineral lost to tailings occur in the scavenger circuit. A study should be initiated to determine why this happens.

2. When the Smelter has the necessary process in place to handle lower sulphur content concentrates, the Mill should again experiment with regrind of scavenger cleaner feed. Significant nickel recovery improvements may be possible.

3. The characteristics of the rougher concentrate cleaner tailings (RCT) are not well understood. A study should be initiated to determine the best method of treatment for Pn/Po separation.

2. Magnetic Separation

Since the pyrrhotite in T-1 type ores is mostly magnetic, it is not surprising that they work well for pyrrhotite removal from concentrates. Test work has generally shown the following trends for magnetic separation of T-1 ores.

a) They are good Po/rock separators

b) They are poor Po/Pn separators

c) Products from magnetic separators need retreatment.

In general, the higher the nickel in the magnetic separator feed, the higher the nickel content in the magnetic product. Therefore they are efficient in removing pyrrhotite, but at the same time carry a high penalty for nickel lost with the pyrrhotite. Lab and plant experience has shown that they are effective in removing Po from scavenger cleaner concentrates and could be equally as effective on treating scavenger cleaner feed.

Magnetic separation of pyrrhotite from scavenger cleaner feed was plant tested in 1988 (results reported in progress report #20). Mass balances around the circuit showed that 65% of the Po in scavenger cleaner feed was recovered to magnetics product, while 30% of the nickel in scavenger cleaner feed was lost with the pyrrhotite. It should be noted that it was not an ideal flowsheet as more magnetic separation capacity was required (ie: 3 magnetic separators were required while only 2 were available). Nevertheless, this did give a good indication of future potential.

Conclusions

Magnetic separators are an important tool for pyrrhotite removal from T-1 type ores and will continue to be used for pyrrhotite management in the future.

Unless methods are found to improve their Pn/Po separation efficiency, we can expect to lose around 25-30% of the nickel to magnetic product in whichever flotation stream we treat.

Future Study

1. It has not been fully established where magnetic separators should be located in the T-1 ore processing flowsheet. Test work must be conducted on treating; a) primary feed to rougher flotation, b) rougher tailings, c) scavenger concentrate and d) scavenger cleaner feed.

2. Magnetic separation of Pn/Po from future ultramafic content ores must be tested. The Birchtree Mine ore, which is currently escalating in production would be the first priority.

3. Alternative Reagents

A high proportion of the overall project time was dedicated to pyrrhotite depressant reagent studies. The behaviour of Pn/Po separation with soda ash is well established. However, other potential pyrrhotite depressing reagents were brought to our attention by Cyanamid Co. and Inco Copper Cliff who also had SO2 abatement studies in progress. Work conducted on Cyanamid reagents has been previously reported in 1988-1989. Although several of the Cyanamid reagents were established to have Po depressing qualities (based on both lab and plant test work) they were not considered suitable because they also depressed pentlandite. Diethylene Tri Amine (DETA) was found to work very well in Inco Copper Cliff, and consequently was also tested in the Thompson Mill. Lab test results showed that Pn/Po separation is improved in the pentlandite circuit provided that sufficient conditioning time is given. The report on this test work will be forwarded to Canmet as a separate report.

Conclusions

Although considerable effort was spent on Cyanamid reagent testing, no breakthrough on Pn/Po separation was made.

DETA, based on a limited amount of test work does appear to have potential to favourably improve Pn/Po separation.

Future Work

1. Thompson Mill personnel should continue to communicate with the Inco Copper Cliff research people on the DETA test work and

2. Continue laboratory scale test work using DETA on various pyrrhotite rich streams in the T-1 ore circuit.

3. Pn/Po separation of ultramafic ores will also be required in the near future.

4. Column Flotation

A factorial design test campaign was conducted in 1988-1989, varying air, water, frother and collector rates while treating scavenger cleaner feed. A 3' dia. by 40' high, fully instrumental column was used. The detailed design of the column has already been made available to Canmet. In the best test case it was shown that 72% of the scavenger cleaner feed Pn could be recovered while 67% of the scavenger cleaner feed Po was rejected to tailings. This compares to conventional Pn/Po separation of 91% Pn recovery with 38% Po rejection. The detailed report will be sent to Canmet as a separate report.

Conclusions

Column flotation does show potential for pyrrhotite rejection from scavenger cleaner feed. However, the 70% level of pentlandite recovery is not adequate to make this an economic option at this time.

Future Work

1. Further test work should be considered, whereby two or three columns in series are tested. Its possible that if the tailings from the 1st column could be refloated in a 2nd column, equivalent good Po rejection would occur while achieving economic Pn recoveries.

    * Minnovex Technologies Inc. of Toronto have a portable 3 column unit flotation pilot plant which would be perfect for this test work. It is therefore recommended that the Thompson Mill seriously consider budgeting $50,000 - $75,000 for this test work in either 1990 or 1991.

5. Current "A"-Circuit Optimization

During the course of the pyrrhotite rejection project, changes were made in the plant operating circuit aimed at improving the Thompson Mill's pyrrhotite management capabilities and rock rejection efficiency. These changes had both short and long term benefits.

One major change occurred in the scavenger circuit, where improved feed distributors and individual flotation bank froth controllers were installed. The more uniform distribution of feed to the float cells as well as better level controllability of the cells resulted in improved pyrrhotite and rock rejection control. Prior to this, individual cell flooding occurred in many instances and certain flotation banks received as much as twice the feed as other banks.

An 1800 cu. ft. rock/desliming circuit was installed in 1988, which treated the scavenger cleaner concentrate. This significantly improved the filterability of the lower pyrrhotite content concentrates, and thus their process ability in the Smelter.

A flotation circuit mass balance was successfully completed in March-April, 1988. This information was valuable for; a) reinforcing the current Mill knowledge of where the pyrrhotite areas of concern are, and b) assisting with future test work in increasing pyrrhotite rejection (a copy of the mass balance was forwarded to Canmet in progress report #19, August, 1988). The mass balance around the rock/desliming circuit is contained in progress report #20 (December, 1988).

Conclusions

Good mechanical control of the flotation equipment is essential to properly manage pyrrhotite and rock rejection.

As Po rejection is increased, rock rejection must also increase. The rock in the concentrates is a dead burden on the Smelter roasters and furnaces and increases the filterability problems as well as the heat balance problems.

Mass balances around the flotation circuit are essential to improvements in pyrrhotite and rock rejection capabilities.

Future Work

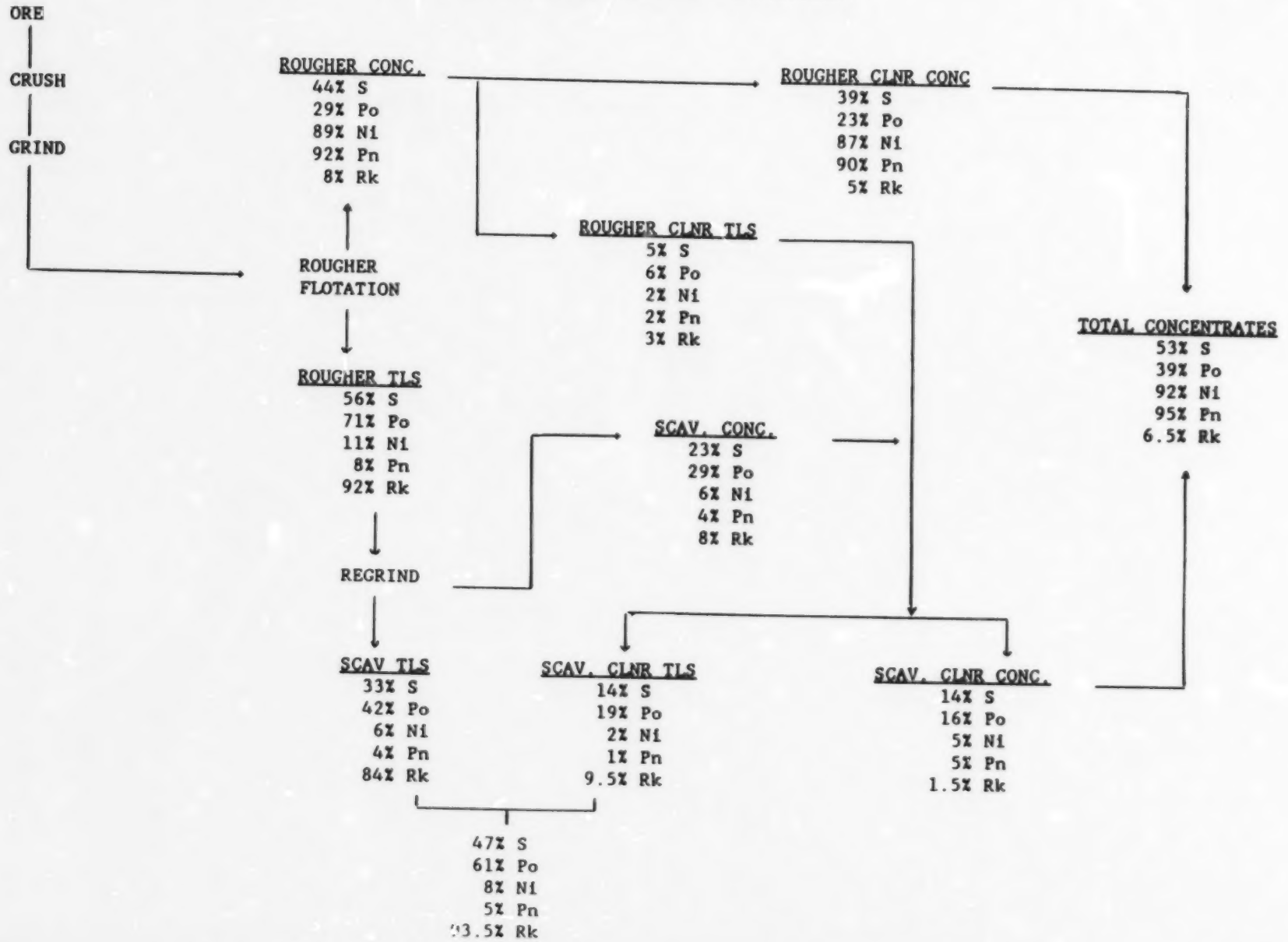
1. Individual flotation bank level controllers should be installed on all the flotation circuits, in a manner similar to the scavenger circuit.

2. A study should be initiated on Pn/rock rejection from concentrates in the rougher concentrate side of the flowsheet. (The majority of the rock in current Mill nickel concentrate comes from the rougher concentrates.)

3. Another flotation circuit mass balance should be conducted during the 1st half of 1990.

PT-284.WPP-4

MILL FLOW SHEET
47% SULFUR REJECTION (SPECIES DISTRIBUTION)



6. Pyrrhotite Rejection from Rougher Concentrates

Approximately 25% of the T-1 ore processing circuit pyrrhotite reports to the rougher concentrates and is associated with 90% of the pentlandite. Because of the high nickel content, Po/Pn separation becomes very costly in terms of nickel loss to tailings. In the long term, the logical flotation stream to treat for Po removal would be the pentlandite concentrate (Sherritt-Gordon concentrate) circuit tailings (referred to in the mass balance as P4). This stream contains around 15% of the overall feed pyrrhotite. The problem is that almost 50% of the feed pentlandite is also in this stream. J.R.G.R.L. were asked to look at this material but were not able to do any research due to other priority work. Thompson Mill personnel did, however, carry out some laboratory scale studies on Po/Pn separation of pentlandite circuit magnetics. Approximately 200 - 300 T/MD of magnetics can be isolated, containing around 5% of the feed Po (2-3% of the feed Pn). Since this material assays around 6% nickel, it is too rich to discard to tailings. Laboratory test work using high SAX rates and lime was able to produce a separation whereby 70% of the pentlandite circuit magnetics Po could be rejected while losing only 30% of the Pn. Test work was reported in 1987 progress reports. More recent laboratory and plant test work, associated with Pn/arsenic separation has shown that pyrrhotite can be isolated from the P4 stream using a combination of pH, SAX and frother addition strategies. However, much more work is required in this area to make this an economical process.

Conclusion

Po/Pn separation on pentlandite circuit magnetics is possible, but a 30% loss of Pn from pentlandite circuit magnetics can be expected.

Po/Pn separation on pentlandite circuit tails (P4) has not been fully explored yet, but projections are that Pn losses would be high.

Future Work

1. J.R.G.R.L. should resume test work on Pn/Po separation of P4 stream.

2. Thompson Mill should carry out more detailed test work on rougher concentrate magnetics.

7. Smelter Energy Balance

Dr. R. G. Orr has coordinated an extensive study on the Smelter energy balance associated with increased pyrrhotite rejection levels. This study was necessitated as the Thompson Smelter was designed to obtain most of its energy requirements through the oxidation of iron sulfides. At present the Smelter is autogenous at 60 - 65% Po (45-50% Sulphur) rejection levels. The Smelter experiences fuel shortages at pyrrhotite rejection levels of 65 - 70% (50 - 55% sulphur), and would be totally unable to function with its present system at 70-80% Po (55-65% S) rejection levels. As the Mill concentrates become more fuel deficient (increased pyrrhotite rejection), the Smelter's ability to consume internally generated scrap from the converter aisle diminish, to the point where the scrap has to be stockpiled. As it is uneconomical to stockpile this nickel indefinitely, the process becomes unacceptable.

Dr. Orr has determined that some of the long term solutions will require the following:

a) Improved filtering to reduce the heat load on the roasters due to water.

b) Addition of energy to the roasters to ensure adequate tonnage throughput.

b) Carbon addition to the furnaces to provide both good metallurgy, and control on the energy content of the furnace matte,thus ensuring that the heat balance around the Smelter is maintained.

d) Oxygen availability on the converters to improve energy balance thus ensuring scrap is consumed (desireable, but not mandatory).

A detailed report was issued in progress report #19 on August 30, 1988.

In a previous study, reported in progress report #14, April, 1987, Dr. Orr determined that a process for drying roaster feed would cost approximately $16 million/year.

Another strategy that Dr. Orr has been looking at is the separate processing of sulphide anode slimes from the Refinery. The elemental sulphur in SAS currently generates some 20 K tonnes of SO2 with the reversion to the Smelter. With a SO2 limit of 220 K tonnes/year, capture of the SAS sulphur would enable the Mill to operate at a pyrrhotite rejection of around 65-70% (55% S rejection). Dr. Orr has issued a number of reports on this topic. Kilborn Engineering have estimated in detail the cost of one of the several options defined under an earlier contract for definition of processes performed by Alberta Sulfur Research Limited.

Conclusion

One cannot consider SO2 abatement through pyrrhotite rejection in the Mill, without seriously exploring the repercussions created in the Smelter.

In order to process low pyrrhotite content concentrates in the Smelter, major system changes are required. These changes are expected to be very costly and require considerable lead time to implement.

Separate SO2 capture from Refinery sulphide anode slimes would be of great benefit to the Mill and Smelter, in that the Mill could reduce the level of sulphur rejection by approximately 4%, thus improving the Smelter energy balance.

Future Work

1. Dr. Orr is in the process of preparing a final report on Mill-Smelter pyrrhotite rejection study.

2. The SAS processing study is still being investigated.

3. The Smelter is in the process of installing a coke addition system to their furnaces. This system should be in operation early 1990.

4. The Mill will review the list of recommendations generated under this study and schedule a new test program for continuing Pn/Po separation, particularily with ultramafic related ores.

General Summary of Test Program Benefits

The project was successfully completed, exploring all phases of the test work as originally outlined in the scope, and through the experimental knowledge gained has revealed many new avenues to pursue.

Test results from laboratory, pilot plant, and plant scale test work have confirmed projections that in order to achieve a 62% sulphur rejection level with Thompson Mine ore (≈75% pyrrhotite rejection) nickel recoveries will decrease to the 86-87% level. There is some hope, however, that through the strategic use of regrind, magnetic separation and proper combination of reagents such as soda ash and Xanthate that the nickel recoveries could be improved. At this stage of flowsheet development we do not have that process conclusively defined.

Test results demonstrated that the mill can economically operate at a 55-56% sulphur (70% Po) rejection level. Nickel recoveries could be kept at the >90% level.

Test results indicate that nickel losses will increase dramatically if we attempt to increase sulphur rejection beyond the 65% level. The reason for this being that the ratio of Pn/Po of the material being treated is 4:1 (P4 stream).

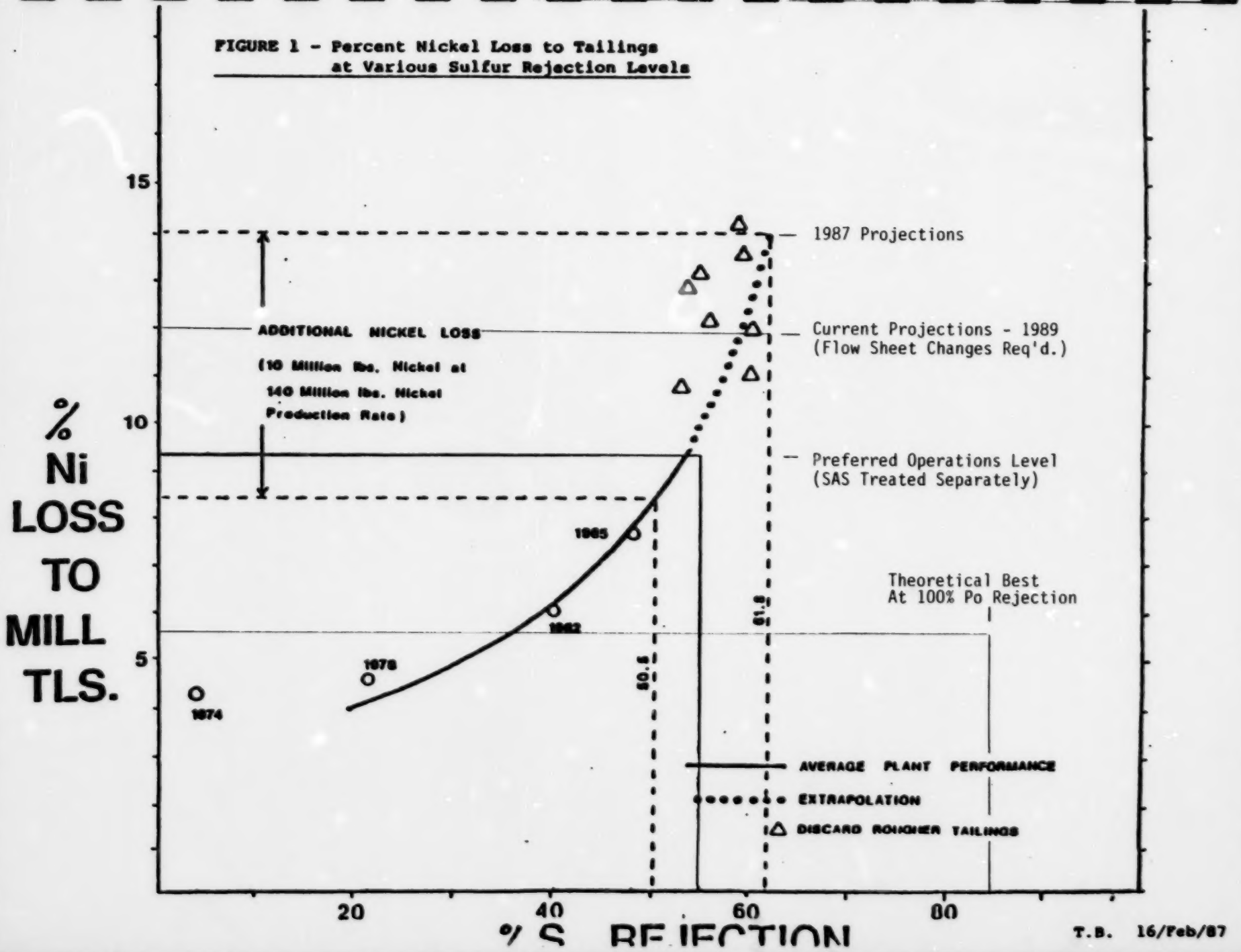
During the course of the project, Smelter concerns with low pyrrhotite content concentrates were identified and studies conducted on possible system changes to cope with the new conditions. Alternative methods for handling Refinery SAS were identified and studies begun to evaluate their feasibility.

Project Duration and Costs

The project which began in August, 1985 was initially scheduled for completion in March, 1988. Due, however, to other project commitments in the Thompson Mill, as well as delays in getting column flotation test work scheduled, the project was extended by around 20 months. The physical test work from the Mill point of view (as per project scope) was completed in June, 1989. The Smelter heat balance study is generally complete but many specific details still remain unconfirmed. The SAS treatment study is still ongoing.

The project consumed the budgeted $338,864.00 with costs charged mainly for Thompson Mill project personnel, but some dollars also charged to Smelter Process Technology and J. Roy Gordon Research personnel in Toronto. It is my sincere belief that excellent value was gained per dollar spent.

FIGURE 1 - Percent Nickel Loss to Tailings at Various Sulfur Rejection Levels



T.B. 16/Feb/87

Final Footnote

As previously mentioned, the above data is valid for Thompson underground (T-1) and Thompson Open Pit (TOP) ores. There is genuine concern on our part that many of these techniques will not work as well on future ultramafic ores. It is therefore imperative that representative samples of future ore bodies be obtained at the earliest possible date so that test work can begin on Po/Pn/rock separation.

Acknowledgements

1. We would like to thank Canmet (Mineral Sciences Laboratories, Department of Energy, Mines and Resources) and especially Mr. Bill Craigen for the project funding and excellent cooperation and understanding throughout the project duration. We trust that the project data generated and reported met with their high standards.

2. We must also recognize Dr. Gordon Agar (J.R.G.R.L.) for his assistance in phase 1-2 and 3 test work.

3. Added thanks to Mr. Dick Alcock in the J.R.G.R.L. exploration department for the mineralogical study.

4. The Mill Process Technology study team is very pleased with the manner in which the project was conducted, and has gained a high degree of job satisfaction from the abundance of project results.

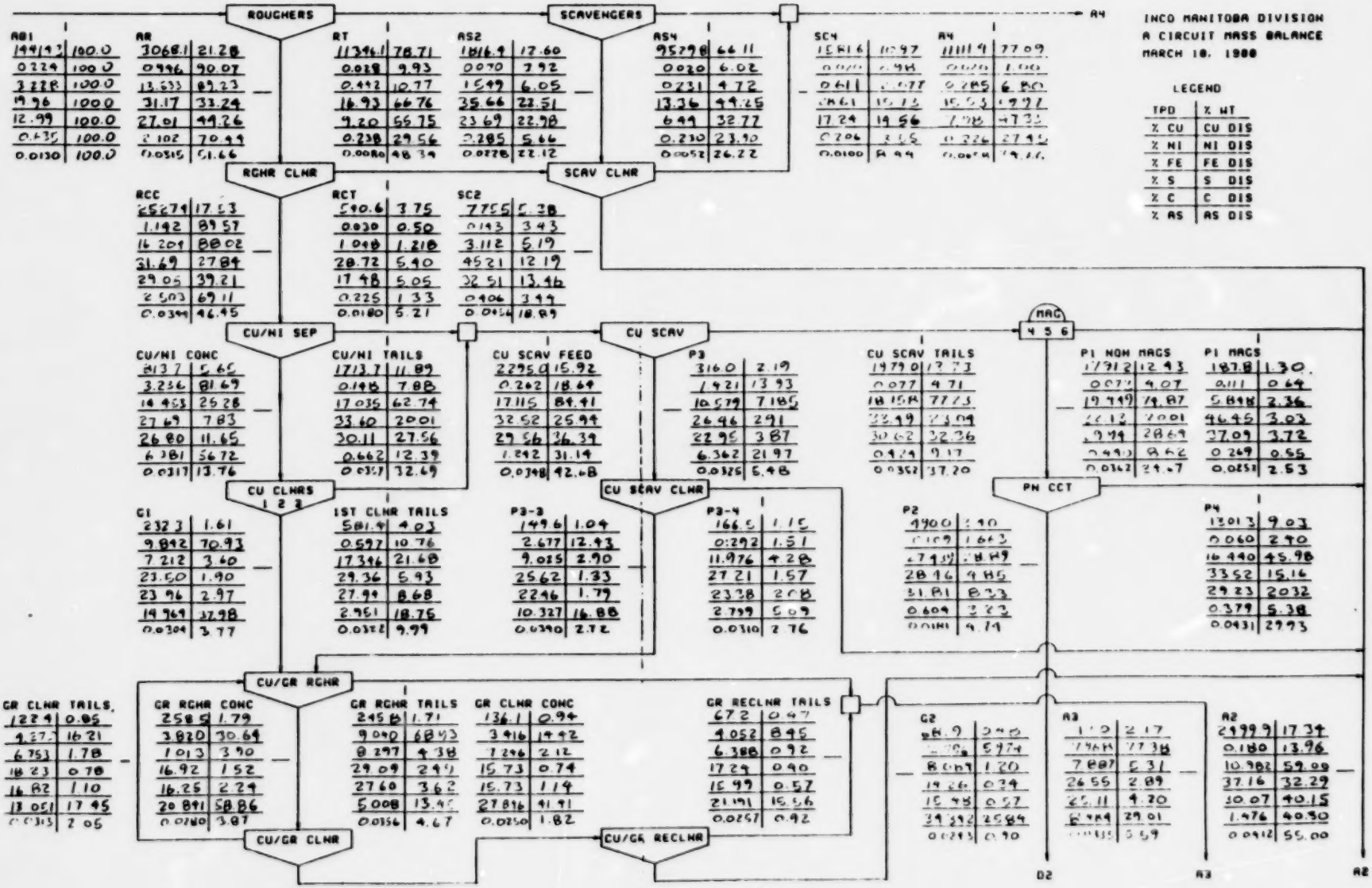
APPENDIX 1

MASS BALANCE OF CURRENT THOMPSON MILL FLOTATION CIRCUIT (MARCH 10, 1988)

Mass balance results compare closely to previous (1984 - 1985) mass balances, indicating that the flotation behaviour of T-1/TOP type ores is consistant and predictable. The balance also gives a good overview of the Pn/Po/rock distribution.

GLOSSARY OF MASS BALANCE DESIGNATIONS

AB1	Combined Mine Ore Feed
AR	Rougher Concentrate
RT	Rougher Tailings
RCC	Rougher Cleaner Concentrate
RCT	Rougher Cleaner Tailings
Cu/Ni Conc.	Copper - Nickel Separator Concentrate
Cu/Ni Tails	Copper - Nickel Separator Tailings
G1	3rd Copper Cleaner Concentrate - Main Mill Copper Product
1st Clnr Tails	1st Copper Cleaner Tailings
GR Clnr Tails	Graphite Circuit Cleaner Tailings
GR Rghr Conc.	Graphite Circuit Rougher Concentrate
GR Rghr Tails	Graphite Circuit Rougher Tailings
GR Clnr Conc.	Graphite Circuit Cleaner Concentrate
GR Reclnr Tails	Graphite Circuit Recleaner Tailings
G2	Final Graphite Concentrate - Added to Nickel Concentrate
AS2	Scavenger Circuit Concentrate
AS4	Scavenger Circuit Tailings
SC2	Scavenger Cleaner Circuit Concentrate
SC4	Scavenger Cleaner Circuit Tailings
A4	Combined Scavenger - Scavenger Cleaner Tailings
Cu Scav. Feed	Copper Scavenger Circuit Feed
P3	Copper Scavenger Circuit Concentrate
P3-3	Copper Scavenger Recleaner Circuit Concentrate (Cu Rich)
P3-4	Copper Scavenger Recleaner Circuit Tailings (Ni Rich)
Cu Scav. Tails	Copper Scavenger Circuit Tailings
4-5-6 Mags	Pentlandite Circuit Magnetic Separators
P1 Non Mags	Pentlandite Circuit Feed (Pyrrhotite Depleted)
P1 Mags	Magnetic (Pyrrhotite Rich) Product
P2	Pentlandite Concentrate For Sherritt Gordon
D2	D2 and P2 are the same
P4	Pentlandite Circuit Tailings
A3	Final Mill Copper Pumped to the Thompson Smelter
A2	Final Mill Nickel Concentrate Pumped to Thompson Smelter


ROUGHERS
SCAVENGERS
A4
RGHR CLNR
SCAV CLNR
CU/NI SEP
CU SCAV
MAG
4 5 6
CU CLNRS
1 2 3
CU SCAV CLNR
PN CCT
CU/GR RGHR
CU/GR CLNR
CU/GR RECLNR
D2
A3
A2
INCO MANITOBA DIVISION
A CIRCUIT MASS BALANCE
MARCH 10, 1980
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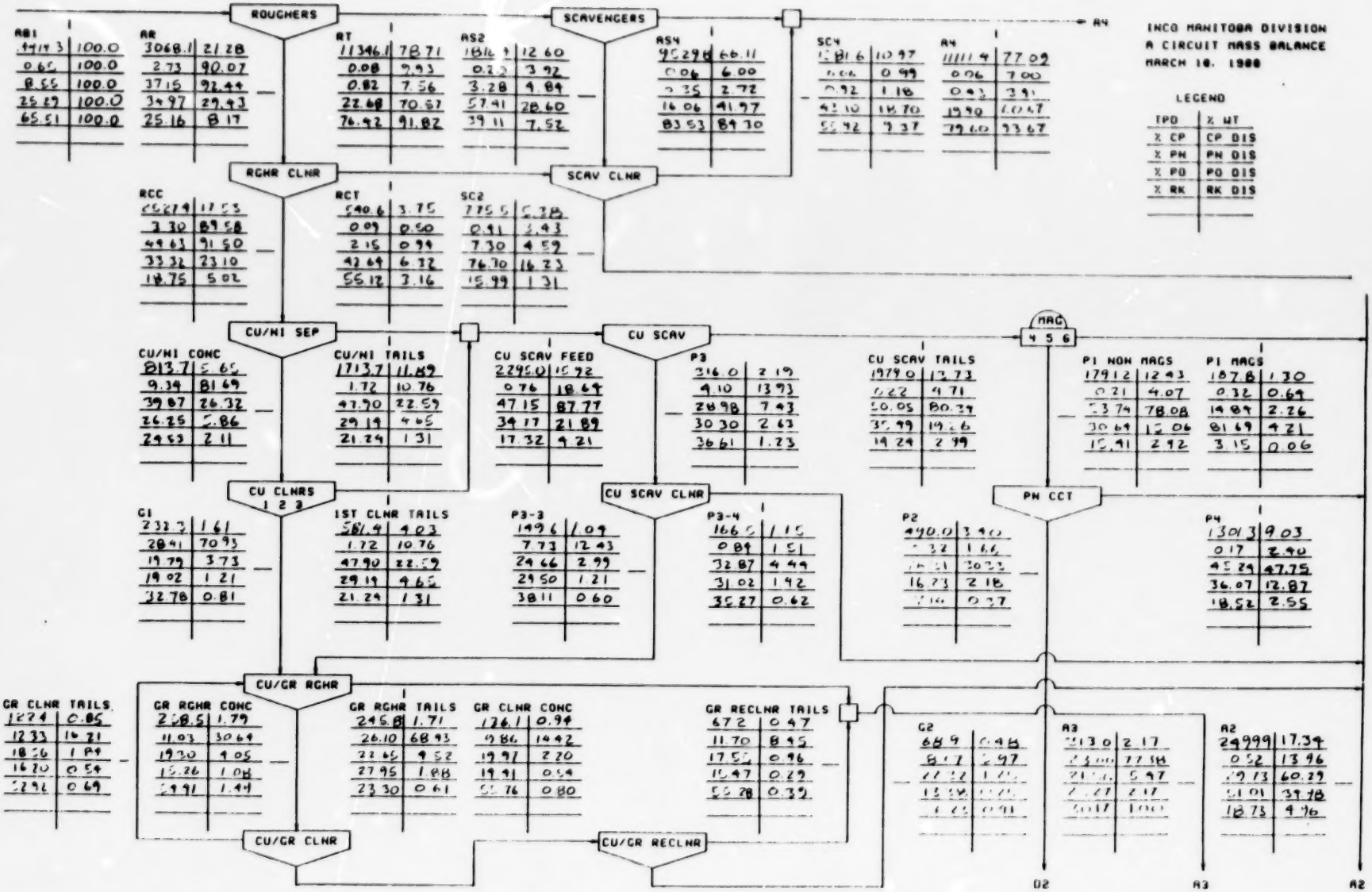
% AS | AS DIS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CU SCAV FEED
P3
CU SCAV TAILS
P1 NON MAGS
P1 MAGS
CU CLNRS
1 2 3
CU SCAV CLNR
PN CCT
C1
1ST CLNR TAILS
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APPENDIX 2

SIMPLIFIED OVERVIEW OF CURRENT MILL OPERATION

Pn/Po/S/ROCK BALANCE AT A 47% SULPHUR

REJECTION LEVEL

FIGURE 1 MILL FLOW SHEET
47% SULFUR REJECTION (SULFUR/NICKEL BALANCE)

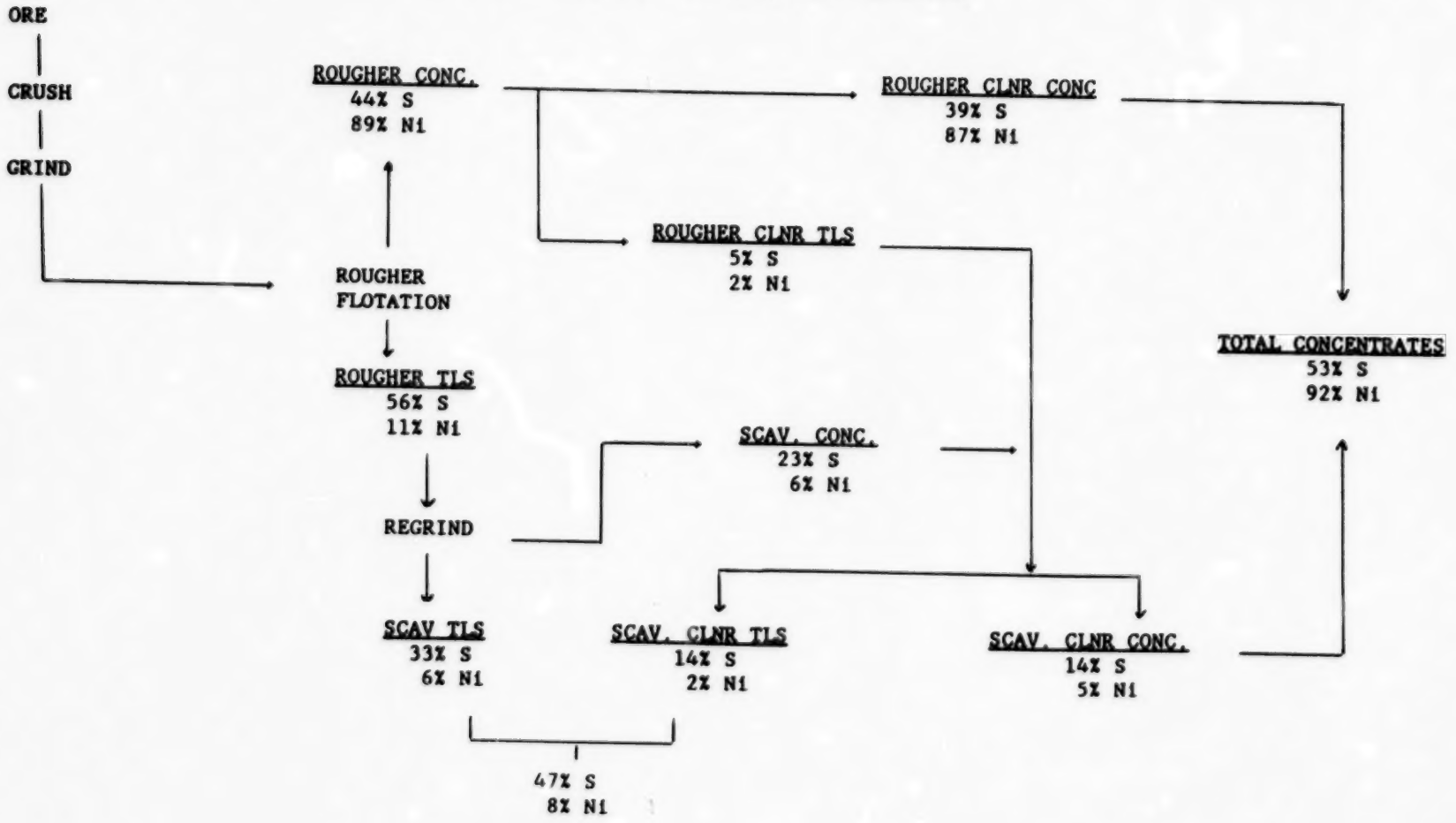


FIGURE 2 MILL FLOW SHEET
47% SULFUR REJECTION (PYRRHOTITE/PENTLANDITE BALANCE)

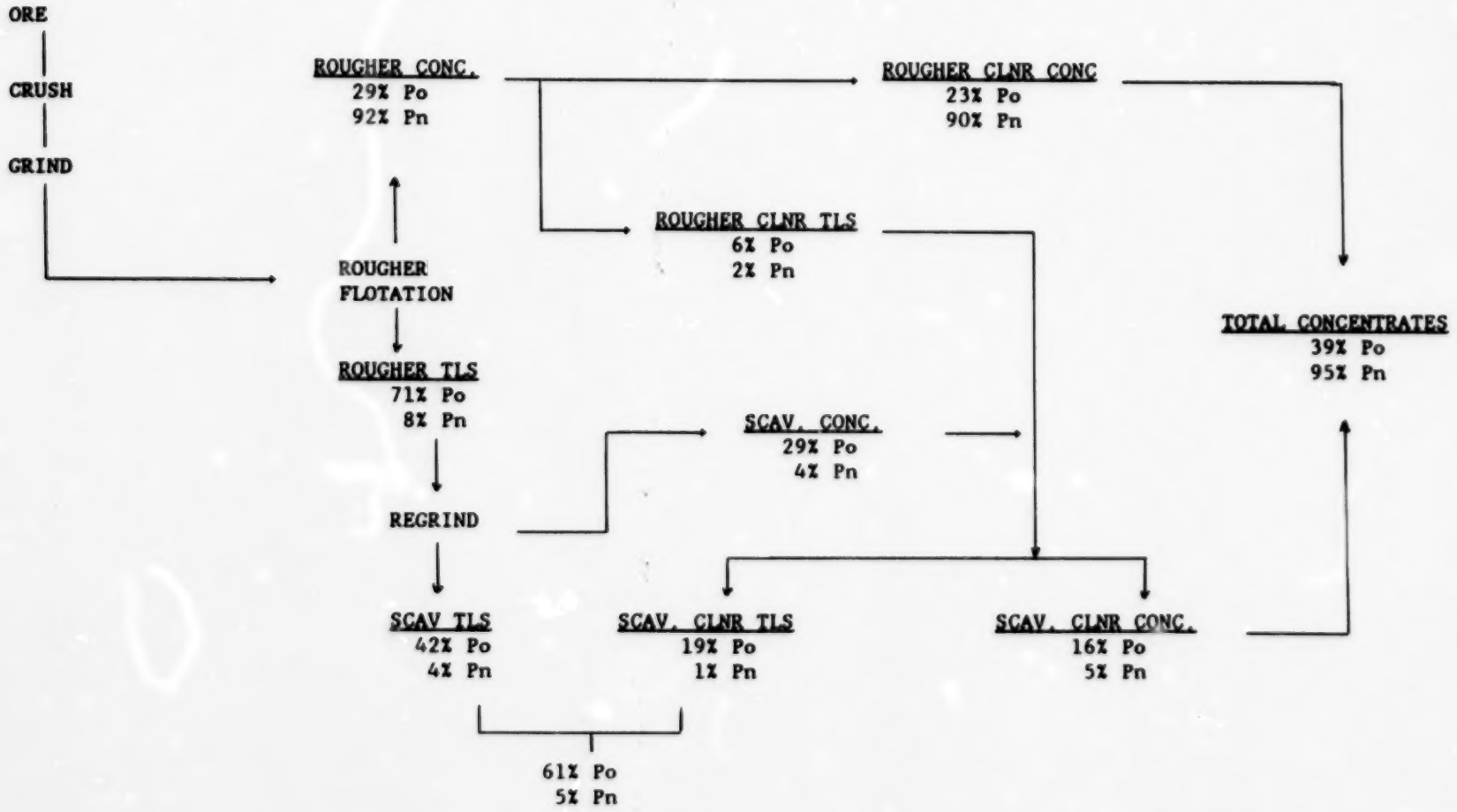


FIGURE 3 MILL FLOW SHEET
47% SULFUR REJECTION (SULFUR/PYRRHOTITE BALANCE)

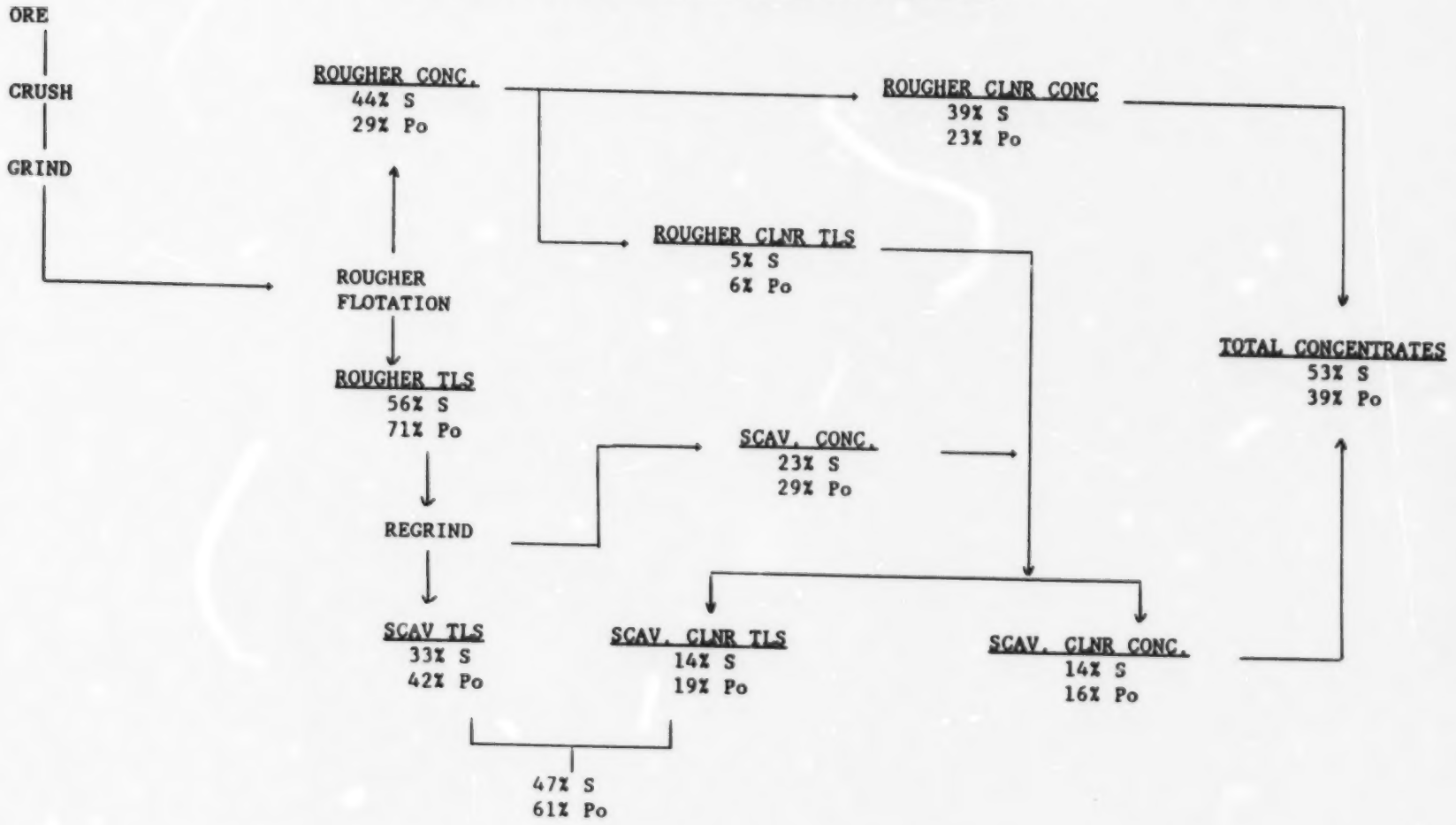
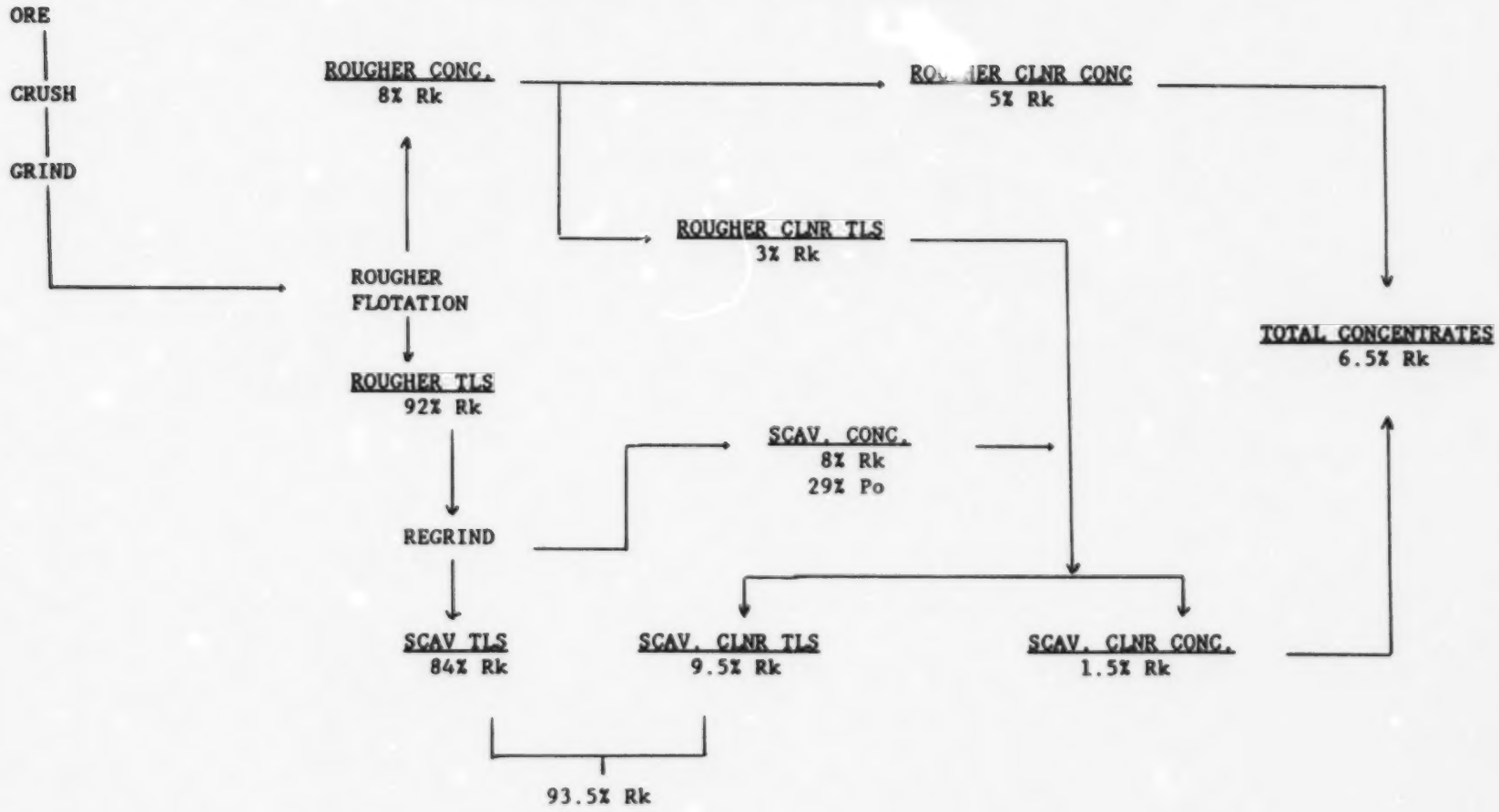


FIGURE 4 MILL FLOW SHEET
47% SULFUR REJECTION (ROCK REJECTION BALANCE)



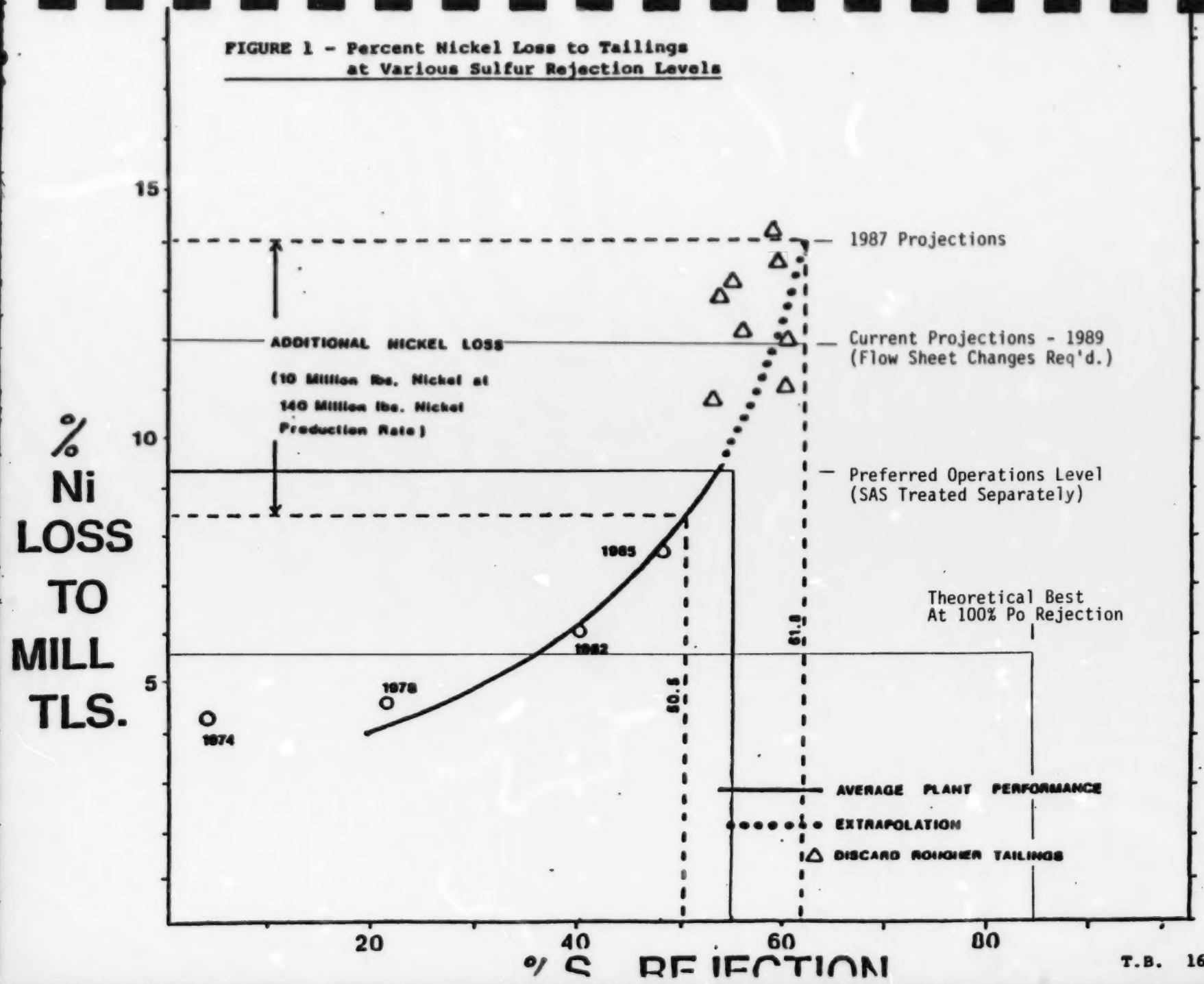
APPENDIX 3

NICKEL LOSS VS SULPHUR REJECTION CURVE

FOR CURRENT THOMPSON MILL FLOTATION CIRCUIT

Test work showed that although some potential exists for decreasing nickel loses, the graph at this point in time is still valid.

FIGURE 1 - Percent Nickel Loss to Tailings at Various Sulfur Rejection Levels



Current distribution of Ni/S/Po in concentrates at 47% S rejection level.

	Distribution		
Stream	% Ni	% S	% Po
Cu Scav. Tls (Pn circ. feed)	77	32	19
Scav. Cleaner Conc. (SC2)	5	14	16
Cu Scav. Cleaner Lbs. (P3-4)	4	2	1.5
Cu Scav. Cleaner Conc. (P3-3)	3	2	1.5
3rd Cu Cleaner Conc. (G1)	3	3	1.0
Total	92	53	39

Future S Rejection Potential

Current - 47% S Rejection at 8% Ni Loss

Discard SC2 - 61% S Rejection at 13% Ni Loss

P3-4 - These streams are impractical to treat.
P3-3 Combined Po only 4.0% of Mill feed.
G1

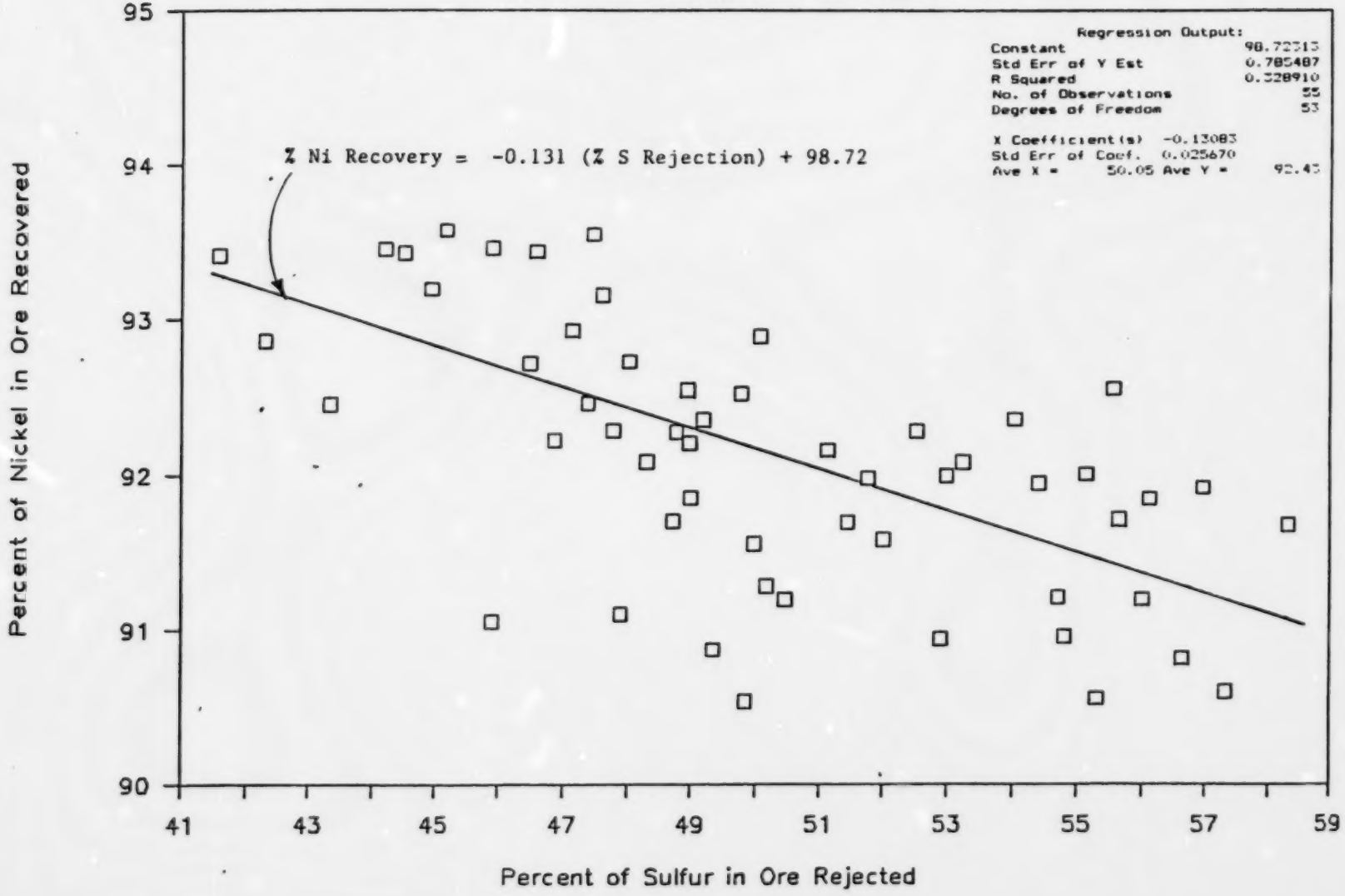
Cu Scav. Tls. - S:Ni ratio of stream is .42:1 almost 50% of the contained sulphur is associated with the pentlandite mineral.

Magnetic separation of Po (based on lab tests) is not very effective in separating Po/Pn.

Area for future work.

Nickel Recovery vs Sulfur Rejection

Total Conc. Weekly Data for 1988



APPENDIX 4

MEMO BY R. G. ORR ON "MEETING THE CHALLENGE OF THE PREDICTED HIGHER FURNACE MATTE GRADES"

1ano2

W. J. Aldag

June 3, 1988

R. G. Orr / I. Farr

Meeting the Challenge of the Predicted Higher Fce Matte Grades

This memo has been written to summarize the authors thoughts on how best the Smelter can meet the challenge of the predicted higher matte grades with the additional accumulation of scrap and probable buildup in the converters.

Recommendations:

As a result of this study we recommend:

- The installation of oil lances in Roasters 1 and 2 as soon as possible.

- The installations to permit the use of tonnage oxygen (10 to 15 tons/day) on the converters be installed expeditiously.

- Capital expenditure estimates be initiated to determine the cost of installation of a quality coke addition system for furnaces 1 and 2 with option for installation on the alternate nickel unit (probably No 4 furnace).

Background:

With the advent of the new SO_2 regulations, the Mill must reject more pyrrhotite from the concentrates to enable the Smelter to meet the emission limits. This will inevitably result in higher furnace matte grades unless some process change is made. High furnace matte grades have two major effects on the Smelter operations:

- The converter aisle cannot consume sufficient scrap to maintain a balance resulting in an accumulation of scrap. With the "normal" variations in furnace matte grade, there are periods where the scrap is consumed but the average results in an accumulation. (Last year estimated to be seven tons/day - and this is estimated to increase to between 15 and 20 with the higher pyrrhotite rejection.) Currently there is about 11 - 12 000 tons of excess converter aisle scrap on hand - and many thousands of tons have been reprocessed by the Mill.

- At matte grades of the order of 38 - 40 % NiCuCo, the converters tend to start to build up accumulations of "mush". This reduces their effective volume and places excessive strain on the mechanical components of the converters. In the past year there have been several incidents where with the normal variations in the furnace matte grade, the Smelter has operated within this critical zone or at higher matte grades. Fortunately the excursions

Background Continued:

into this critical operating zone were of short duration and the following period of low matte grades enabled the converters to digest this buildup. The required higher pyrrhotite rejection levels is predicted to result in the furnace matte grades increasing by approximately 2.5 % on average, increasing both the frequency and duration of excursions of operation at the undesirable furnace matte grades. Figure 1 demonstrates this.

This problem is NOW! Solutions are required NOW! However, the solutions should be compatible with long term operating plans.

We have examined several options which are briefly discussed below:

1. Add Sufficient Pyrrhotite and Reduce Production

While a viable solution, the current market conditions preclude this solution.

2. Removal of Pumped Refinery Reverts from Roaster Feed

Computer modelling of expected concentrates and reverts indicates this would reduce the matte grade sufficiently to alleviate most of the matte grade problems, especially the converter buildup problem. While this might assist in overcomming a crisis, assuming the anode count was high enough, these reverts must be reprocessed some time (and somewhere) into a form suitable for refining. Virtually every alternative flowsheet for processing the current pumped Refinery reverts involves drying the reverts AND the equipment could not be in and operational in the short term. Drying of the pumped refinery reverts with "separate" smelting facilities may be a long term solution.

3. Oil Lances in Roasters 1 and 2

Installation of oil lances into Roasters 1 and 2 will assist in controlling the matte grade in several ways:

- Reduce the iron sulfide oxidation in the roaster thus pushing more into the furnace matte.

- Permit operation of the roaster at lower average temperatures even when the oil is not required because with adequate roaster controls (available now) the operator would be assured the roaster temperature would not fall below the concentrate ignition temperature. This would result in less iron sulfide oxidation in the roaster on average and hence a lower average matte grade and less buildup of scrap.

Oil lance technology is available now.

The oil lances could be installed within a very short time (1 to 2 months)

3. Oil Lances in Roasters 1 and 2 Cont'd

A disadvantage is the cost of the oil which is estimated at roughly $ 500 000/yr.

4. Addition of Fine Coke to the Furnaces

The addition of coke to the furnaces will lower the matte grade, add energy efficient metallic iron to the furnace matte and give superior metallurgy in the furnaces. The major stumbling block in the past was the furnace vents, but with the refractory vents now installed on furnaces 1 and 2, this is not a problem. The major problem is adequate control of the quantities of carbon added. Preliminary studies in the past indicated the cost of the carbon would be more than offset by the increased recovery of metals in the Smelter.

The addition of carbon to the furnaces could be done several ways:

- Via sand system and through the roaster. This has the advantage that the equipment is available now. The disadvantages are primarily:
    - the difficulty of good control of the addition rates
    - the burnoff in the roasters (estimated at 10 - 40 %)
    - possible sintering problems in the roasters
- Via the scrap conveying system currently being installed. This is not a particularly optimal solution as the carbon would not be intimately mixed with the calcines which has been shown to be a requirement in the past.
- Via the current floor sweeping dust system. The major disadvantages to this proposal is the manual labour involved and the problem of control of rate of addition.
- Installation of a coke addition system. This is the preferred long term solution, but it will take extensive capital installations.

5. Preheating of Roaster Blast Air

Preheating the roaster blast air with direct combustion of fossil fuels has little advantage over the use of oil lances directly into the roasters as the effective energy efficiency is the same.

Preheating the blast air with electrical heaters (or indirect heating) would be effective but would require major capital installations and is not available on a short term. Potentially, the transformers for No 3 furnace could be used. A problem with electric roaster air preheating is the current 160 MW limit we have with Manitoba Hydro. There is insufficient power available during the winter months without raising this limit.

4. Use of Tonnage Oxygen in the Converter Blast Air

The addition of tonnage oxygen (approx. 10 - 15 tpd) to the converter blast air would consume all excess scrap, the current scrap pile (over 1 to 2 years) and probably eliminate the converter buildup problem.

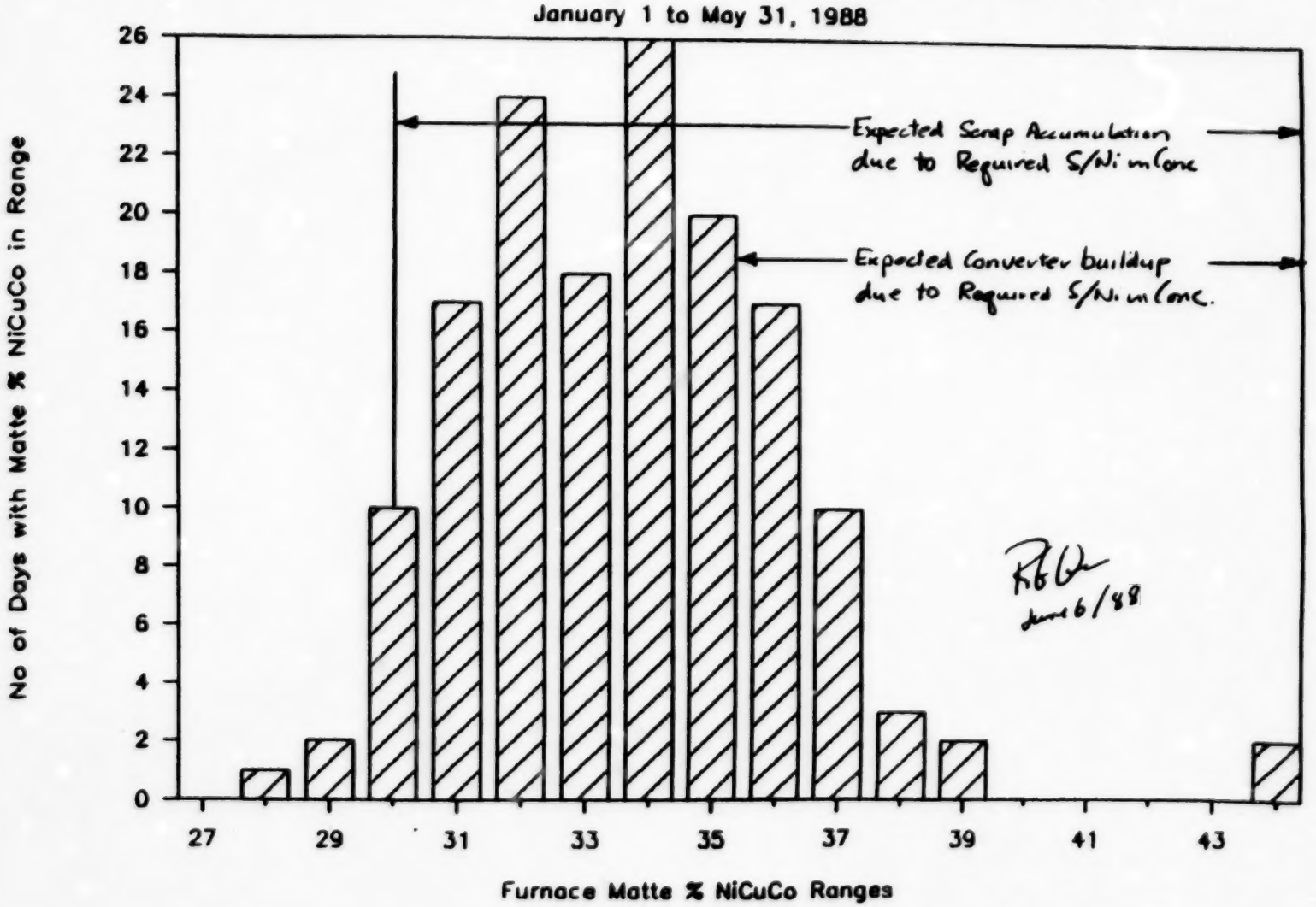
Unfortunately, the installation of tonnage oxygen availability requires extensive capital installations. A capital project to install tonnage oxygen is mandatory for the Refinery and studies on the addition of sufficient capacity to supply the Smelter as well are in progress.

I. Farr

R. G. Orr

cc: R. I. Barton
D. M. Shefford
R. T. Hamson
R. L. Perchaluk
J. C. Greenhill
J. A. Chapman
J. Ashton
File (2)

Figure 1 Distribution of Ni Fce Matte % NiCuCo



APPENDIX 5

FUTURE ORES - ULTRAMAFIC RELATED ORES

By 1995 - 1996, up to 50% of the Thompson Mill feed is expected to be ultramafic in content. These ores behave much differently than current Thompson type ore.

spmpm2

To: D. M. Shefford

July 3, 1989

From: R. G. Orr

Behaviour of Ultramafic Ore: Pipe Lake Ore Milling, 1983 - 4

Introduction:

Much of the Manitoba Division Ore Reserves are classified as being ultramafic. The Thompson Ore body is not ultramafic. Pipe Lake Open Pit was an ultramafic type ore. The main ore body at Birchtree Mine is classified as ultramafic although most of the early (i.e. first two years) production from the refurbished mine will not be classified as ultramafic.

Most of the development work on pyrrhotite rejection for SO_2 emissions control has been done on Thompson ore. This is due mainly to the unavailability of fresh ultramafic ore from appropriate ore deposits for testing. While Pipe Open Pit Mine was operating, the level of sulfur rejection from the ore was well in excess of that being achieved with the Thompson Mine ores. The sulfur/nickel ratio in the Pipe Lake Open Pit source concentrate was significantly below the Thompson source concentrates.

The level of sulfur rejection from Pipe Lake Open Pit ores is not directly comparable as the Thompson Mine ores averaged approximately 4 lbs sulfur per lb of nickel and the Pipe Lake Open Pit Mine ores averaged close to 7.5 lbs sulfur per lb of nickel.

In an attempt to understand the significance of the new (and revitalized) ultramafic type ore source mines on the use of pyrrhotite rejection to control SO_2 emissions, the 1983 - 4 Weekly Mill data on the "C" or Pipe Mine circuit was re-examined. This memo discusses the major metallurgical implications.

Pipe Lake Ore:

The nickel content (ore grade) of the Pipe Lake Open Pit Mine ore was quite variable. Figure 1 shows a run chart of the weekly average ore nickel content for 1983 - 4. This shows rapid variations with a range of nickel contents ranging from 0.55 to 1.48 %. A histogram showning the distribution of the average weekly ore grades is given in Figure 2.

The sulfur/nickel ratio in the ore appears to be very much a function of the grade of the ore. This is shown in Figure 3. Increasing the grade of the ore, increases the sulfur/nickel ratio. This implies the higher grade underground ore may have a higher sulfur/nickel ratio than the Open Pit had. A similar correlation has not been observed in the Thompson Mine ore.

Milling of Pipe Lake Ore:

Unfortunately for the study into correlations between sulfur rejection levels, concentrate sulfur/nickel ratio and recoveries, the "C" circuit was operated to produce a tailing of approximately 0.2 % nickel, not to meet a target concentrate sulfur/nickel ratio or sulfur rejection level. This mode of operation results in the data shown in Figure 4, Nickel Recovery vs Ore Grade and in Figure 5, Tailings Nickel Content vs Ore Grade.

The Thompson Mine ore average weekly milling data result in a good correlation between the nickel recovery and the level of sulfur rejection (Figure 6). The parallel data for Pipe Lake Open Pit ore milling results in the data of Figure 7. This data shows no correlation between nickel recovery and sulfur rejection. (the linear correlation coefficent, $r^2 = 0.006$). The distribution of nickel recoveries for the Pipe Lake Open Pit ores is shown in Figure 8.

As a result of the operation of the Mill on Tailings assays, there is no significant correlation between the resulting concentrate nickel content and the ore grade (Figure 9). The concentrate grades (% Ni) were widely distributed as is shown in Figure 10. The flotation capacity of the mill was sufficiently large as to result in no apparent correlation between concentrate and tailings compositions (Figure 11). In fact, the variation in the concentrate nickel content reflects on the variations in the processibility of the ore, and on the stability of the milling process.

The apparent correlation between the concentrate sulfur assay and the nickel content of the ore (Figure 12) is probably a result of a significant decrease in the rock content of the concentrate. This hypothesis is supported by the fact the concentrate sulfur/nickel ratio is not affected by the ore grade (Figure 13).

The distribution of Pipe Mine concentrate sulfur/nickel ratios is shown in Figure 14. The importance of this figure is that it demonstrates the production from lower grade ultramafic type ores of concentrates with sulfur to nickel ratios that would permit Manitoba Division to meet the projected SO_2 emission limits (220 Klonnes/yr) at a net product production rate of 140 million lbs/yr.

Conclusions:

It is possible to produce concentrates from ultramafic type ores (Pipe Lake Open Pit) with a sulfur/nickel ratio at or below that required to restrict SO_2 emissions to below the proposed 220 Klonne limit by pyrrhotite rejection alone. The recoveries of the contained nickel will be a function of the ore grade and probably of the ore sulfur/nickel ratio.

Testwork must be conducted on each potential Mine based on ultramafic type ore to define the expected recovery at the required concentrate sulfur/nickel ratio.

Figure 1 **Run Chart for Pipe Ore Grade**

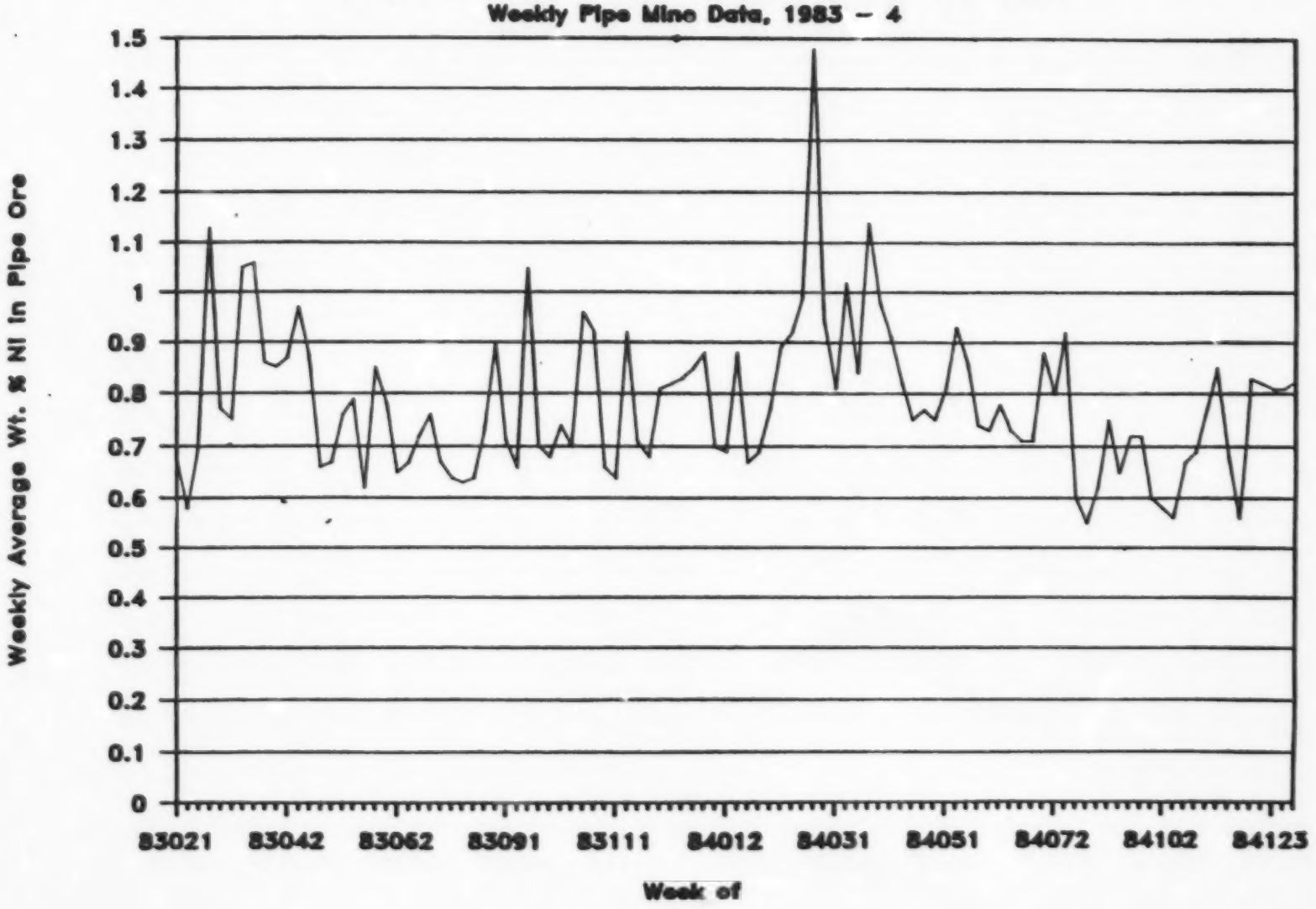


Figure 2: Distribution of Pipe Mine Ore Grades

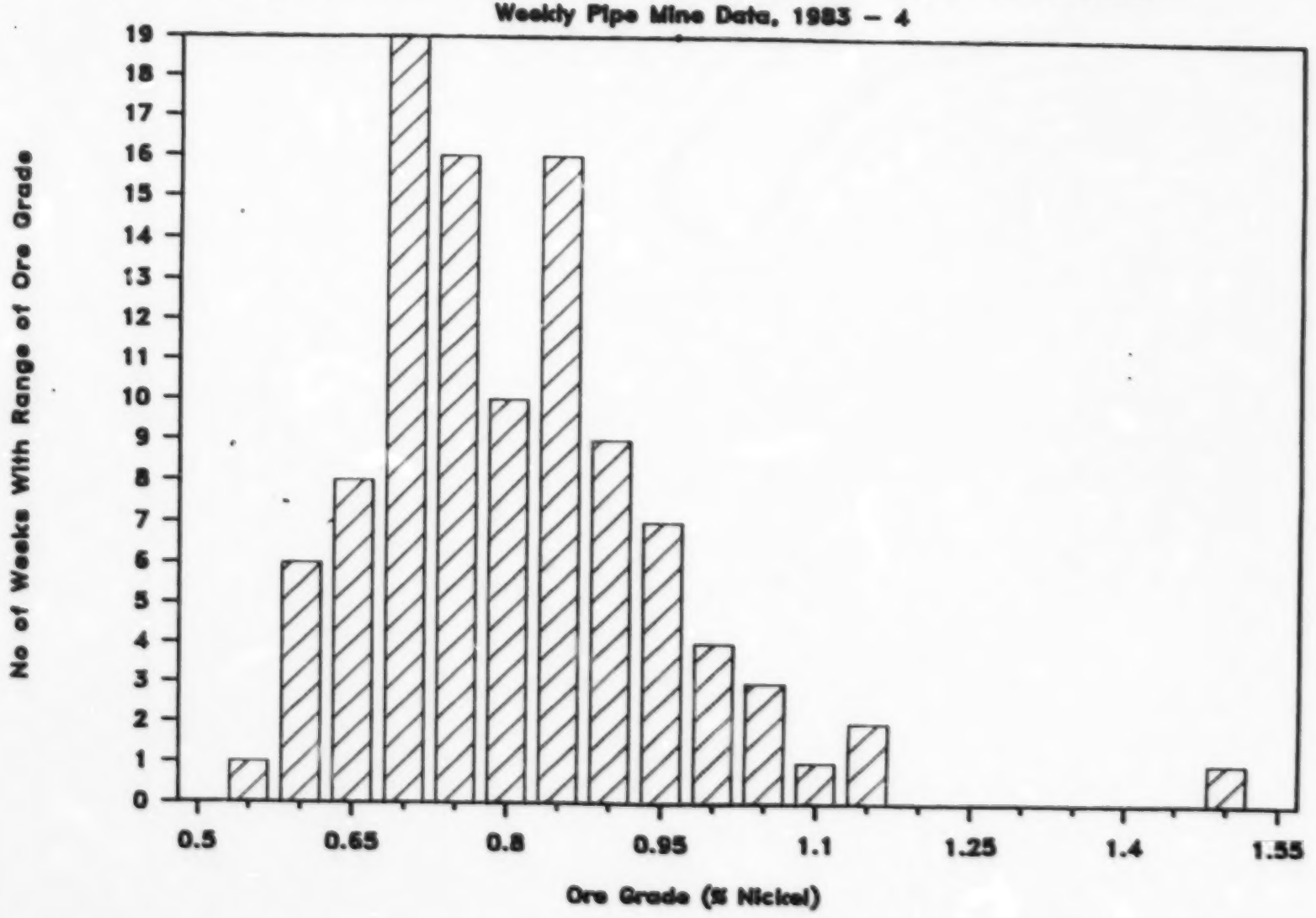


Figure 3: Ore Sulfur/Nickel vs Ore Grade

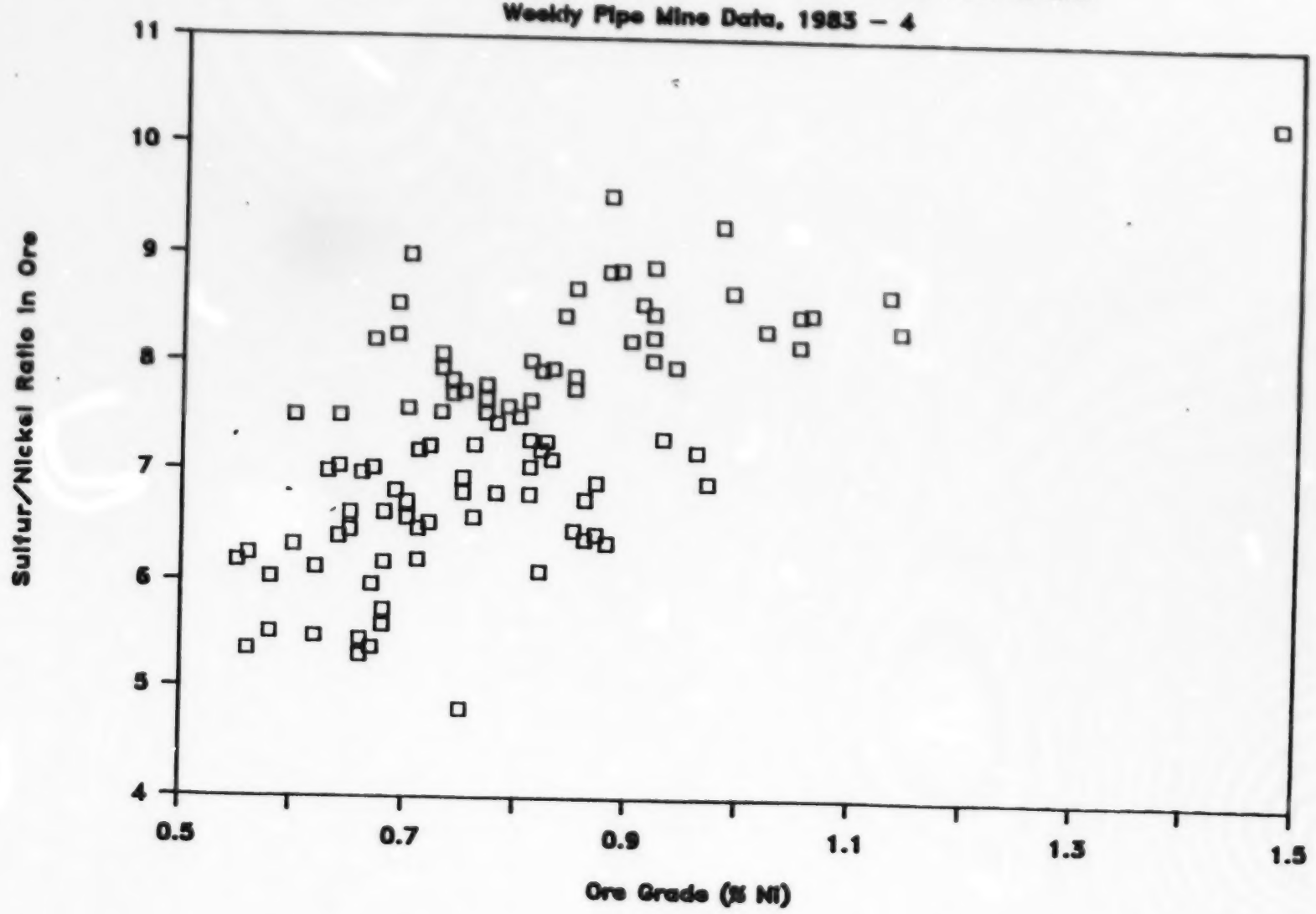


Figure 4: Nickel Recovery vs Ore Grade

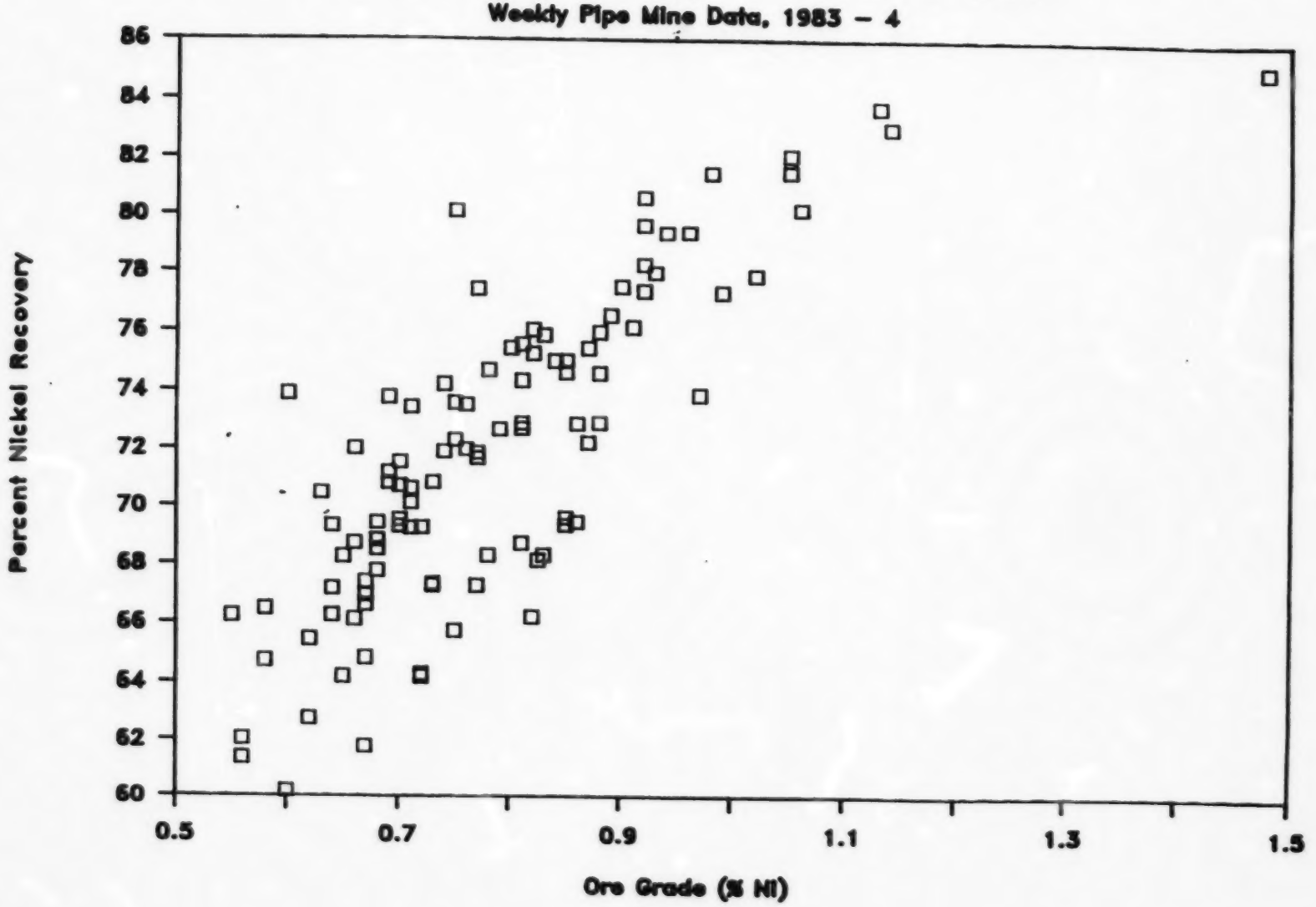


Figure 5: Tailings % Ni vs Ore Grade

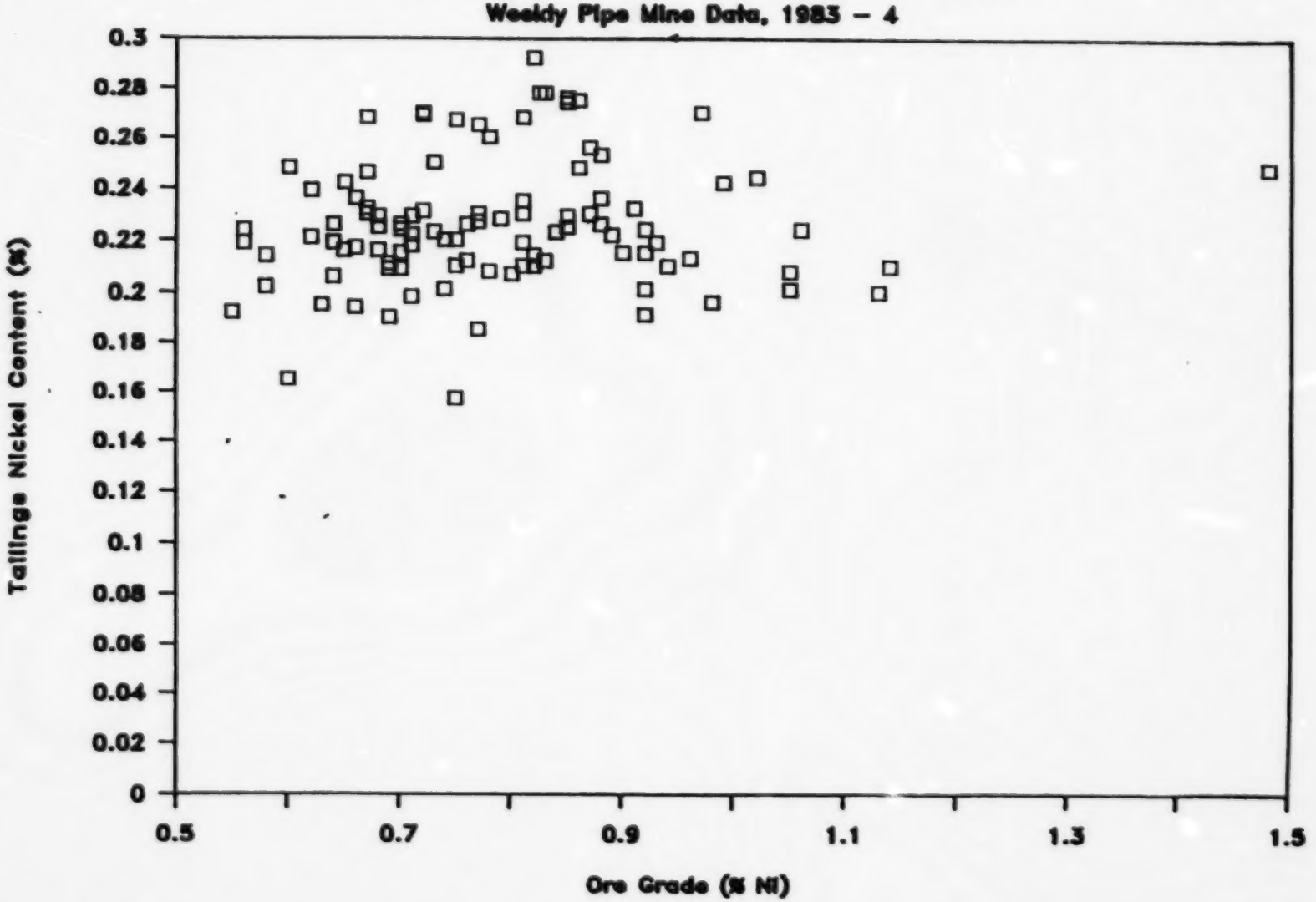


Figure 6: Nickel Recovery vs Sulfur Rejection (Thompson Ore)

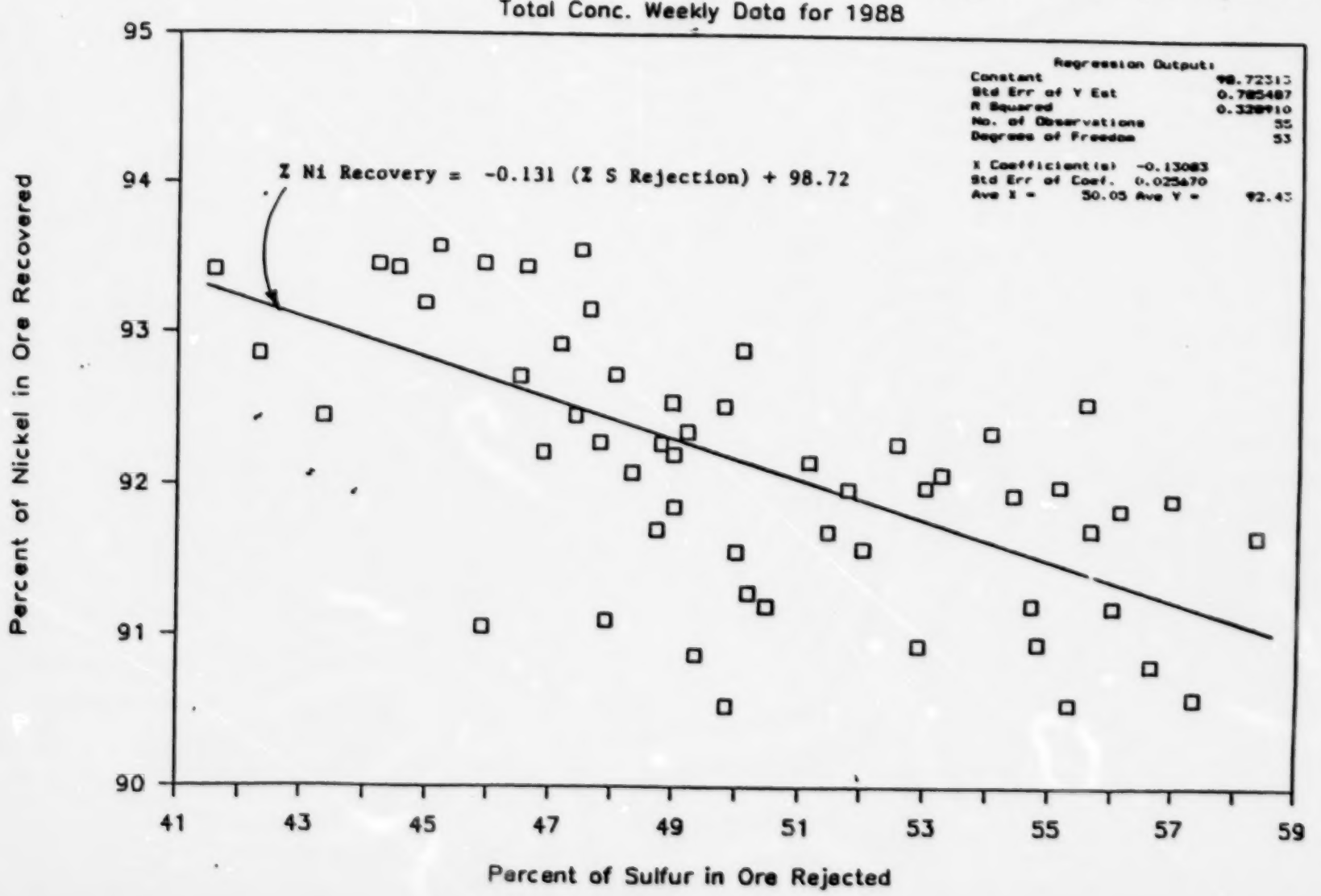


Figure 7: Nickel Recovery vs Sulfur Rejection

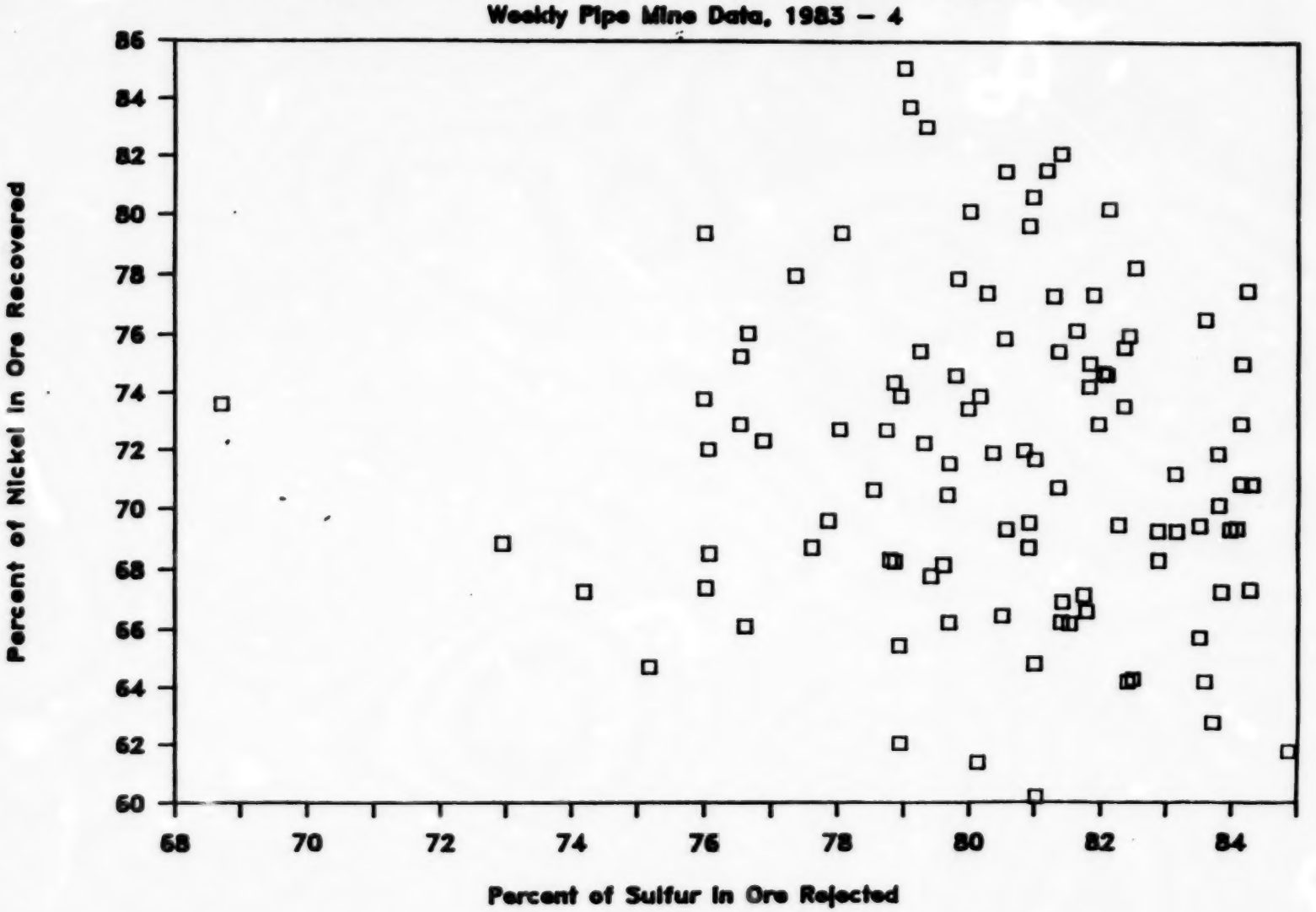


Figure 8: Dist. Ni Recovery to Pipe Mine Conc.

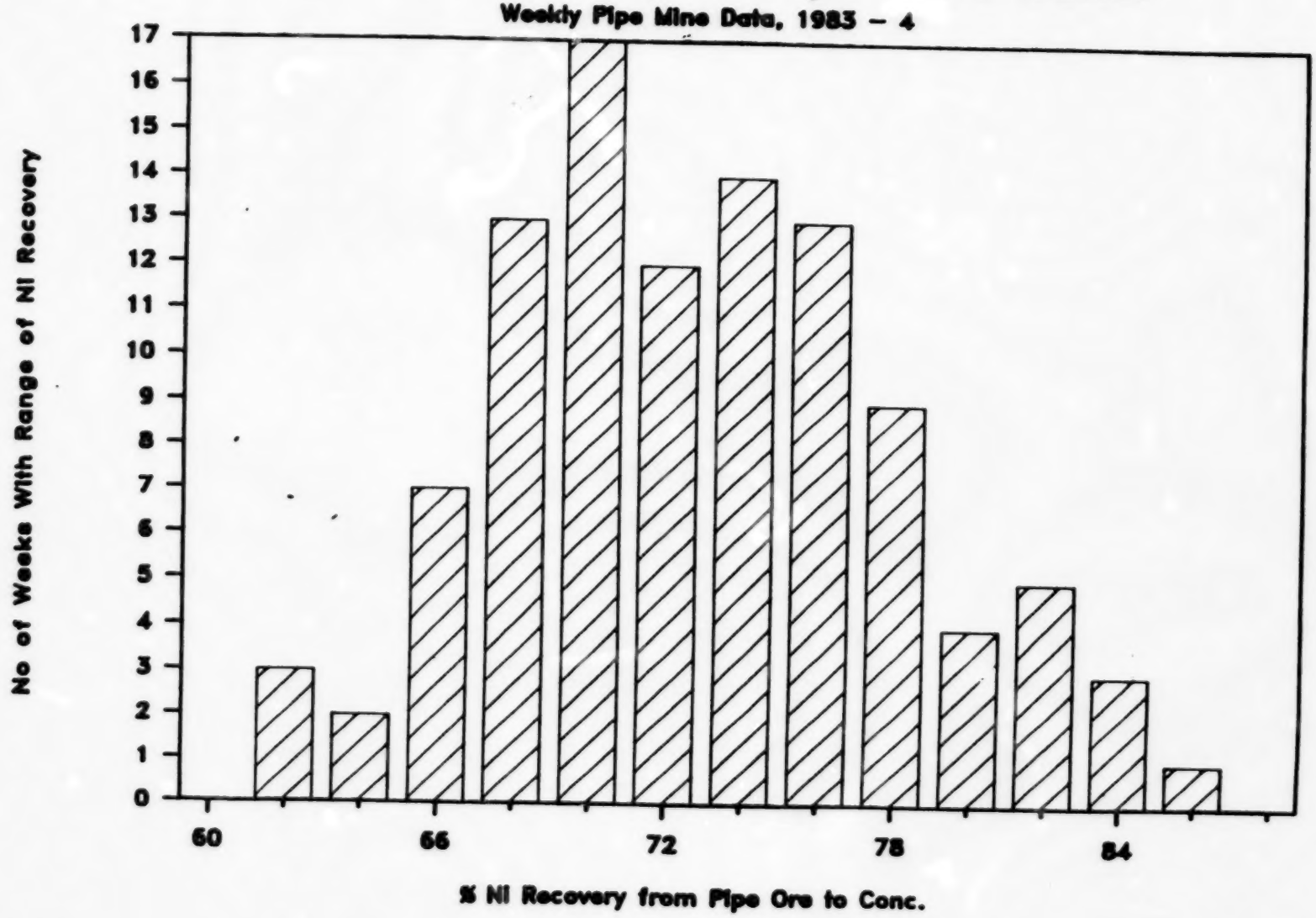


Figure 9: Conc. % Ni vs Ore Grade

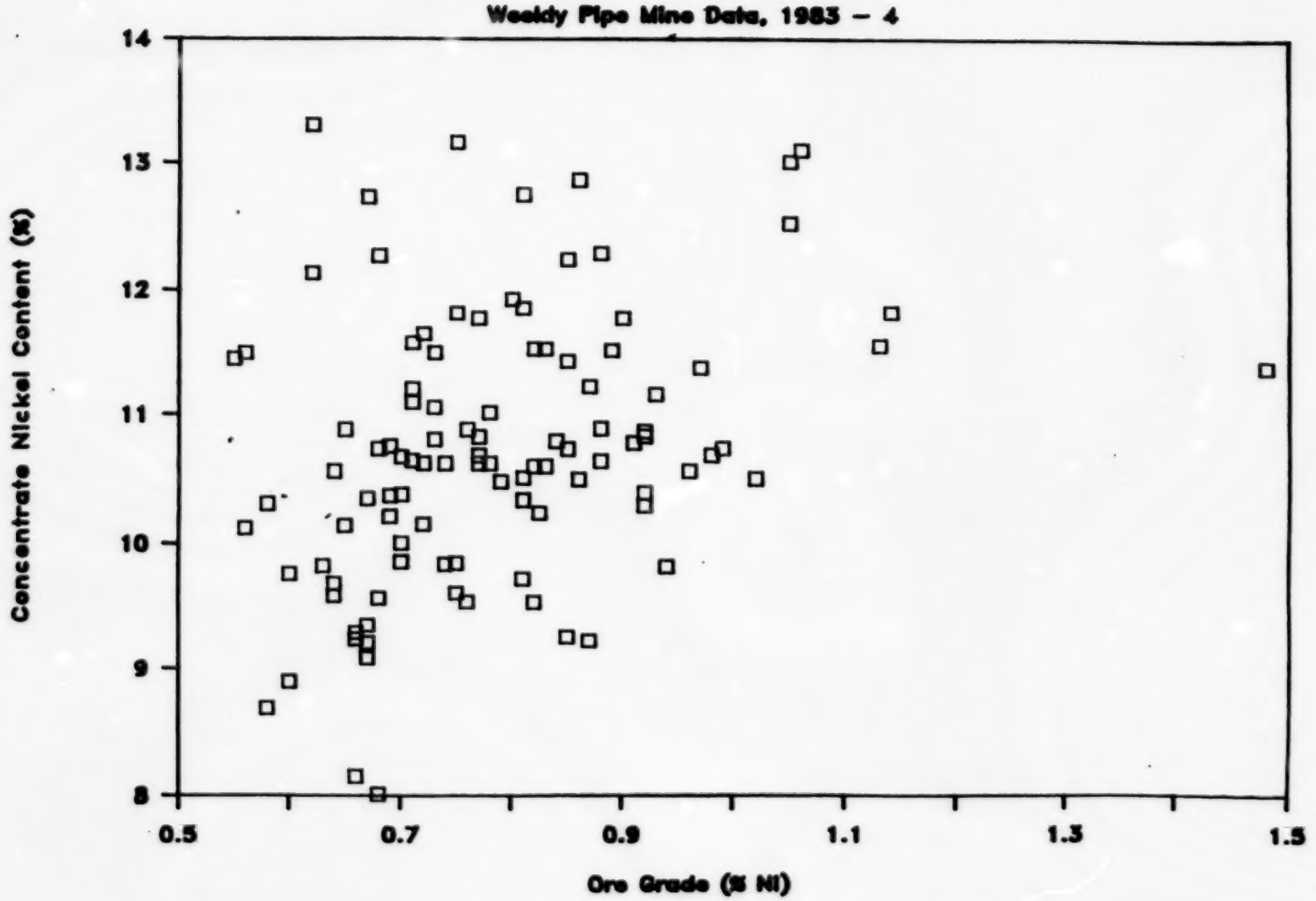


Figure 10: Distribution of Pipe Mine Conc. Grades

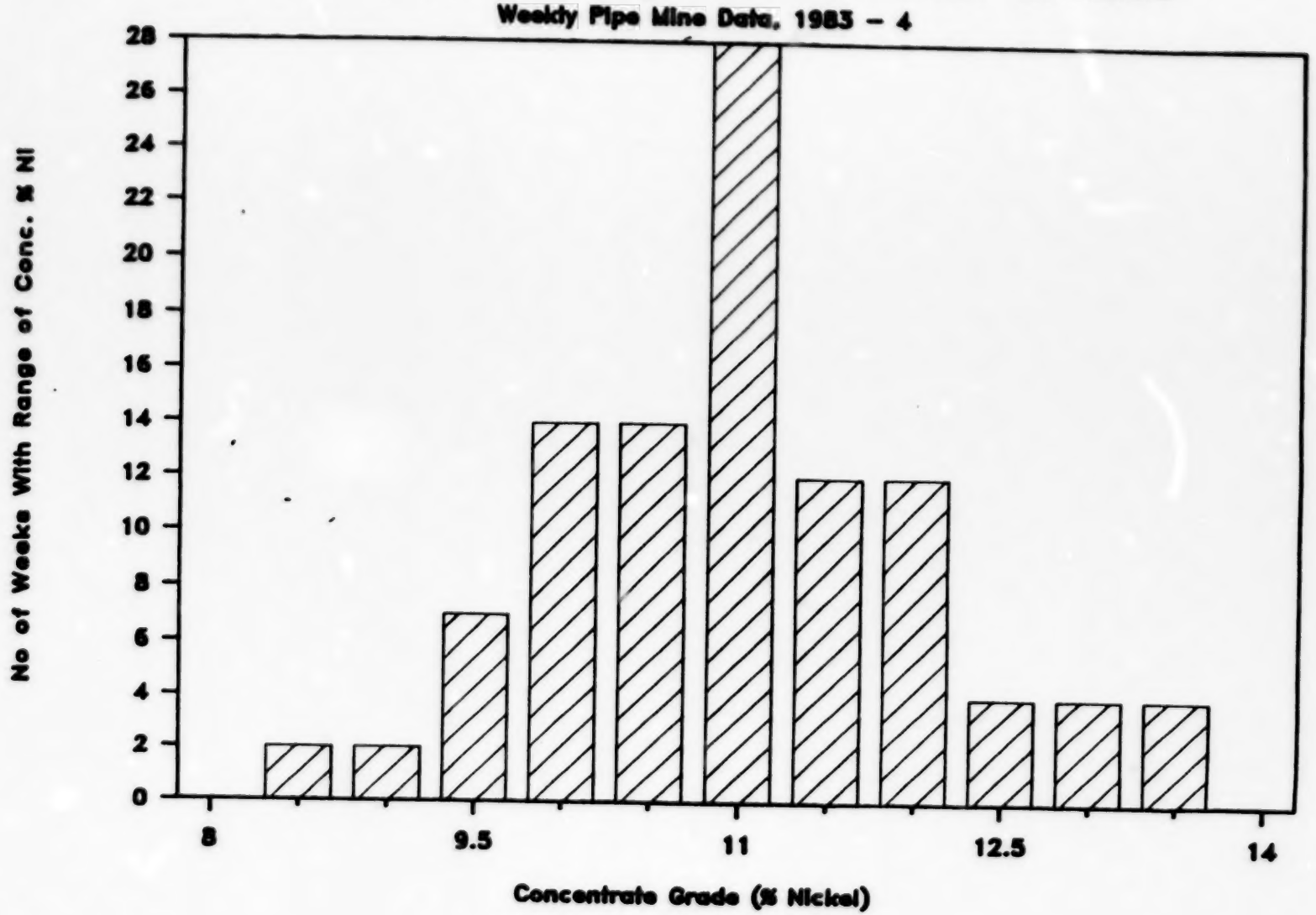


Figure 11:

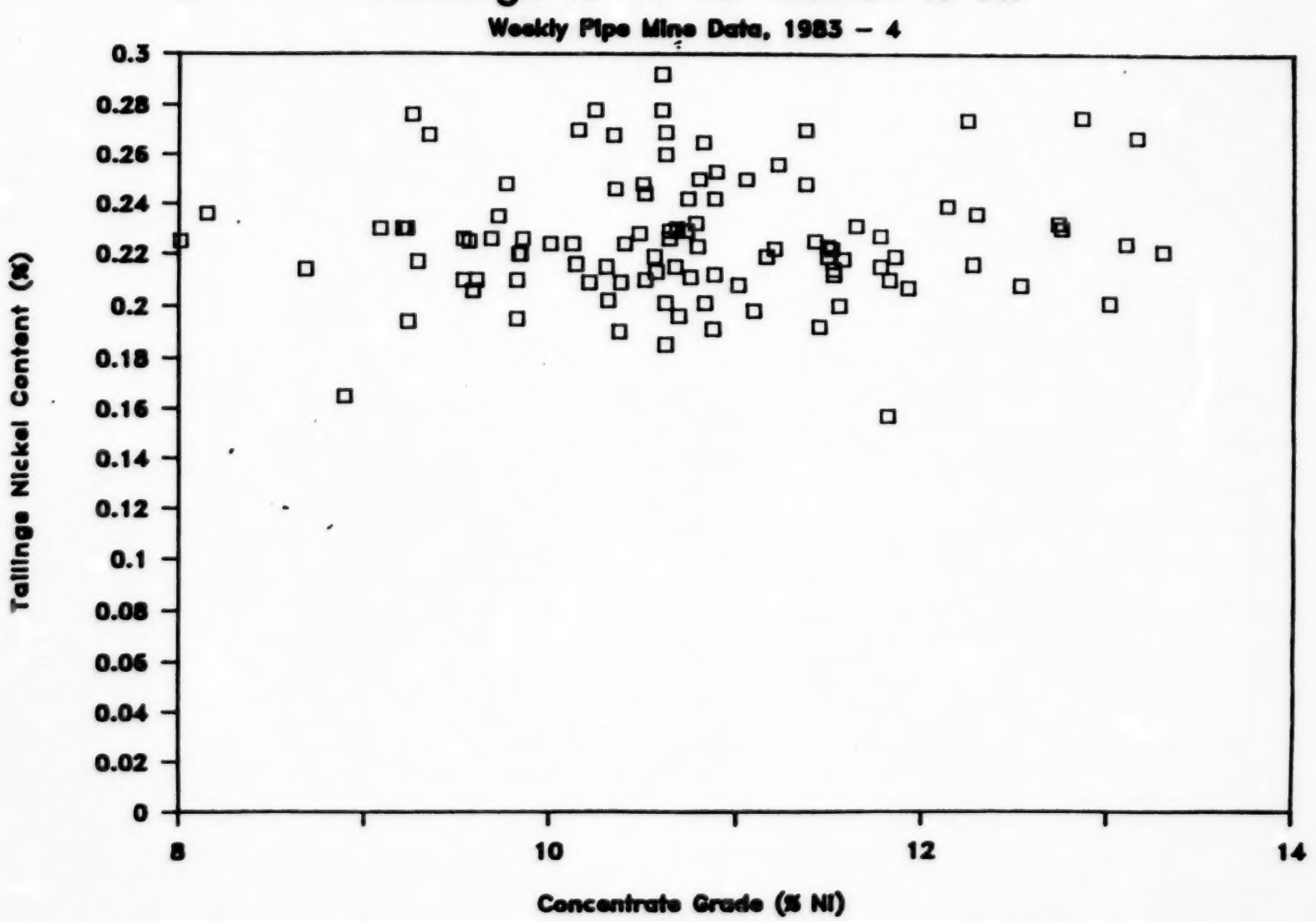

Tailings % Ni vs Conc. % Ni
Weekly Pipe Mine Data, 1983 – 4
Tailings Nickel Content (%)
0.3
0.28
0.26
0.24
0.22
0.2
0.18
0.16
0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
8
10
12
14
Concentrate Grade (% Ni)

Figure 12: Conc % Sulfur vs Ore Grade

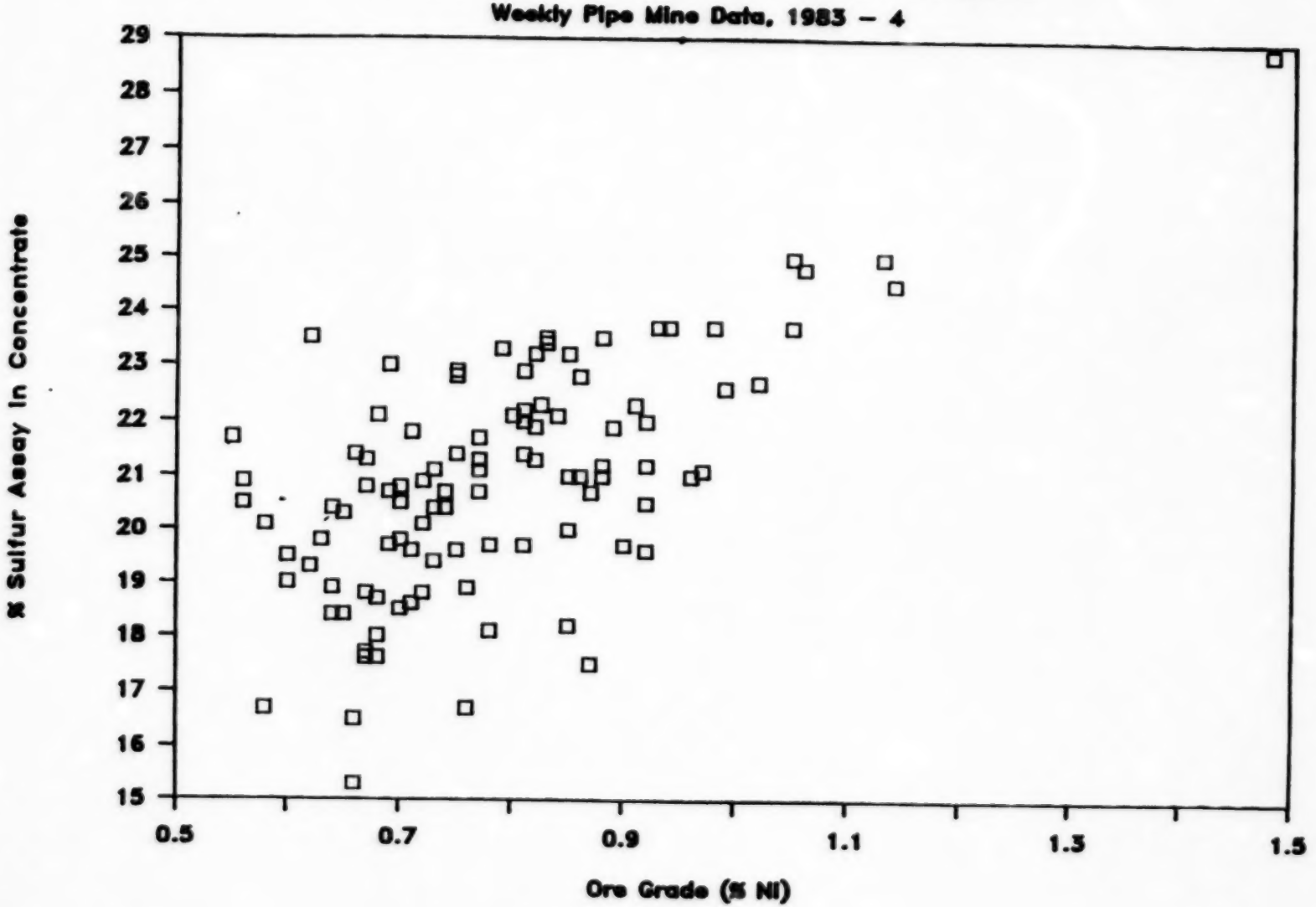


Figure 13: Conc Sulfur/Nickel vs Ore Grade

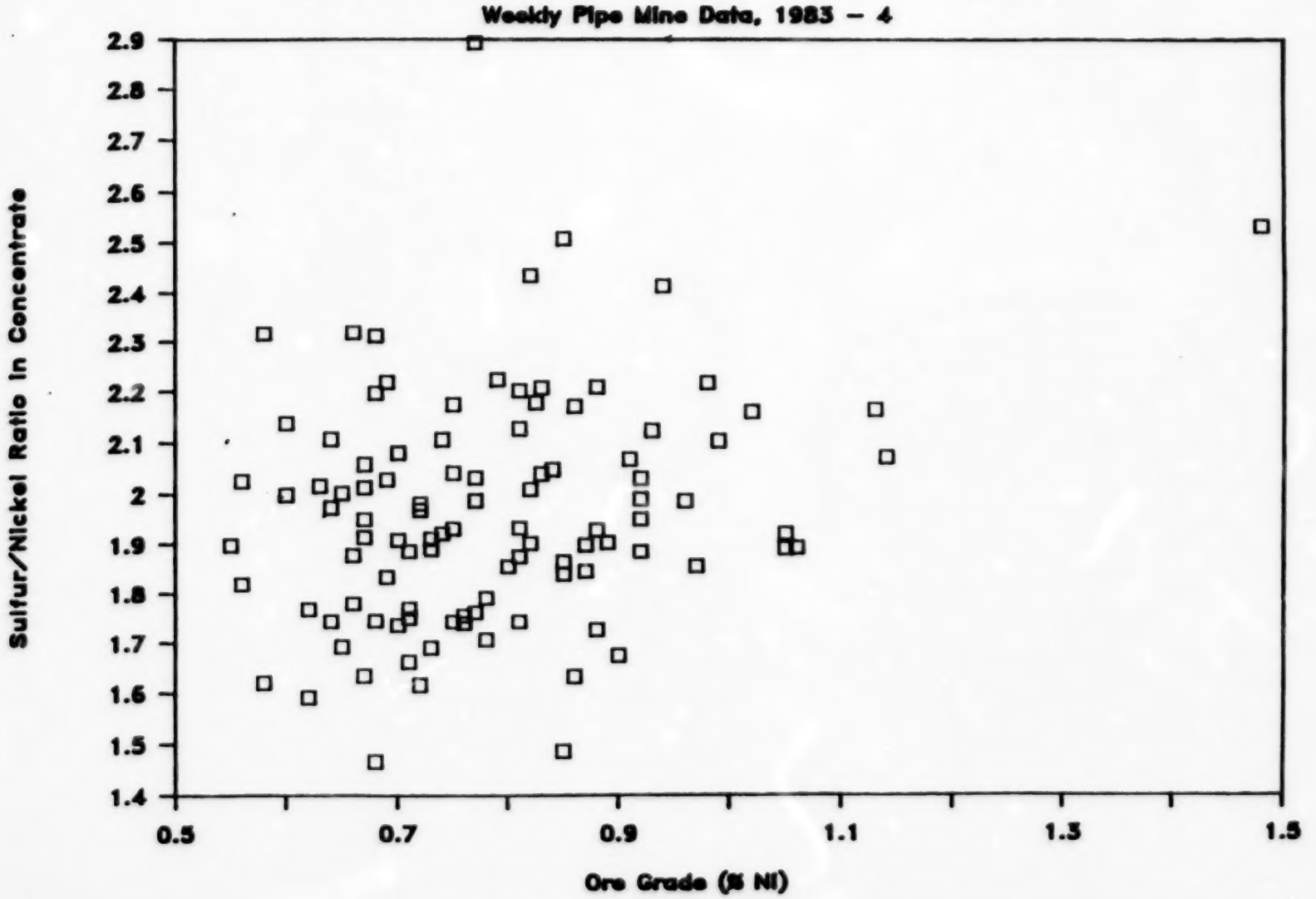
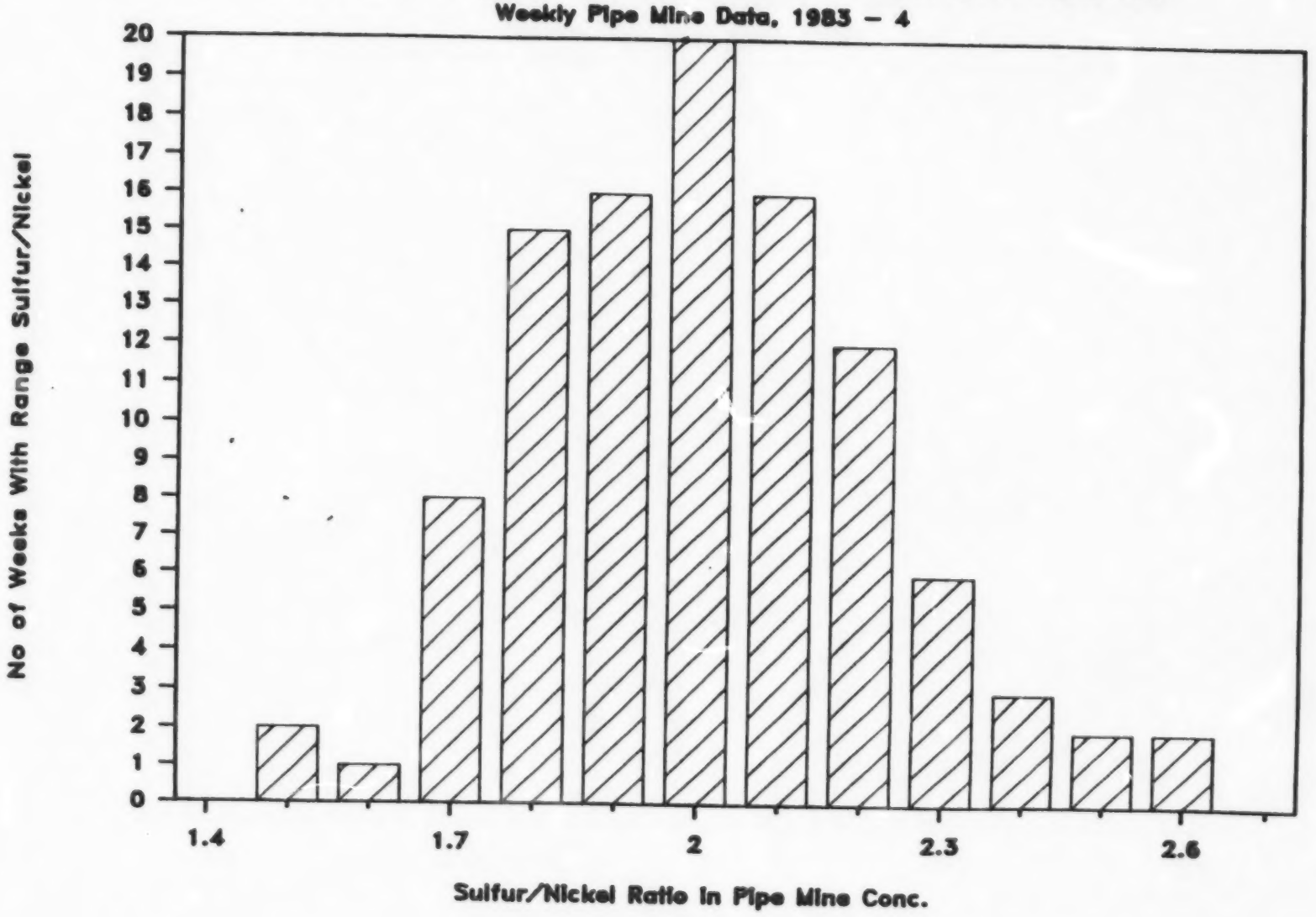


Figure 14: Dist. of Pipe Mine Conc. Sulfur/Nickel



APPENDIX 6

CONTRACT PLAN AND REPORT FORM

FOR 1989 FISCAL YEAR

Supply and Services Canada | Approvisionnements et Services Canada

CONTRACT PLAN AND REPORT FORM – FORMULE DE PROJET DE CONTRAT ET DE RAPPORT

CONTRACT TITLE – TITRE DU CONTRAT

CONTRACTOR – ENTREPRENEUR

1989 Fiscal YEAR.

08BQ.23440-5-9030	08QB5-00112

	1987														
	1 APR.	2 May	3 June	4 July	5 Aug.	6 Sept.	7 Oct.	8	9	10	11	12			
Phase I - Sonovoque Required Testing															
Phase II - Magnetic-Separator Eval.															
Phase III - Reagent Evaluation															
Phase IV - Column Flotation Evaluation															
Phase VA - A Circuit Design After Phase II															
Phase VB - A Circuit Design After Phase III															
Phase VC - A Circuit Design After Phase IV															
Phase VI - Rougher Conc. Re Study															
FINAL REPORT															
													PLAN		